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A STUDY OF THE PROCESSING OPTIONS FOR GOLD RECOVERY OF CAPE SPENCER ORES, NEW BRUNSWICK

by

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A STUDY OF THE PROCESSING OPTIONS FOR GOLD RECOVERY OF CAPE SPENCER ORES

FINAL REPORT

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SUMMARY

This project was designed to investigate the recovery of gold from heap leach pregnant solution. The project included a state-of-the-art literature survey, bench-scale leaching tests, a column leaching test, and bench-scale testing of identified recovery processes. Also, the project included preliminary process economics for a 250,000 tonne per annum heap leach operation.

The state-of-the-art literature survey of heap leach technology and gold leaching showed that potential gold leachants were cyanide, thiourea, thiocyanate, chlorine and chlorine salts, and bromine and bromine salts. Zinc precipitation (Merrill-Crowe), carbon adsorption, ion exchange, solvent extraction and direct electrowinning were identified as potential recovery processes.

The sample used in the testing was a low grade gold ore obtained from the Road Zone of Mispec Resources gold property on Cape Spencer, New Brunswick. The head assay was 0.62 g/t. A mineralogical examination of samples from various areas of the property indicated that the ore consisted of quartz, illite, ankerite, rutile, pyrite and feldspars. Muscovite, chalcopyrite, galena, hematite, magnetite, sphalerite, Cu/Fe sulphates, bornite, covellite, digenite and carrollite were present in some samples. Gold generally occurred in association with pyrite.

Bench-scale testing determined that the sample was readily leached with NaCN. The mineralogical examination followed by bench tests showed that the ore contained a high acid consuming component. Testing with thiourea further confirmed that a low pH milieux is not suitable for this ore.

The column test confirmed that with a nominal top size of 25 mm this ore is easily leached with 2 g/L NaCN solution. Recovery after 40 days was 100% into a pregnant solution assaying 1.5 g/L gold. Cyanide consumption was approximately 1 kg/tonne.

Bench-scale tests of the various recovery processes showed that carbon adsorption and solvent extraction gave the highest recoveries (at least 97%). Zinc precipitation was relatively inefficient at the low solution gold loading; recovery was low and zinc dust usage was high. Direct electrowinning gave recoveries of approximately 70% and ion exchange was not tested because of problems obtaining a suitable resin.

Preliminary capital costs for a 250,000 heap leach operation using carbon adsorption as the recovery process are $1.4 million (± 30%).

RESUME

L'objectif de cette projet était rechercher à l'echelle du laboratoire quelques techniques pour l'extraction de l'or à partir de solutions cyanurées. Le projet compris une revue de litterature, essais à l'echelle du laboratoire, un simulation en une colonne d'un lixiviation de tas, essais de les techniques de récuperacion et les economiques préliminaires pour une plante de 250,000 t/a.

La revue littéraire a indiqué les cinq techniques suivants à récupérer l'or de solutions cyanurées: la cémentation (Merrill-Crowe), le charbon activé, les résines, l'extraction a solvant, et l'électro-extraction.

Le minerai a employé in cette étude venu de le "Road Zone" de le minerai d'or de Mispec Resources en Cape Spencer, Nouveau Brunswick. Le minerai comprends quartz, illite, ankerite, rutile, pyrite, feldspaths, muscovite, chalcopyrite, galène, hematite, magnetite, sphalérite, sulfates de Cu/Fe, bornite, covellite, digenite et carrollite. L'or est souvent associé à pyrite.

Les essais à l'echelle du laboratoire indiquent que la cyanure de sodium a été le plus meilleur lixiviant pour cet minerai. L'essai simulation a confirmé que un broyage de moins de 25 mm, un solution de 2 g/L NaCN et 40 jours de lixiviation pouvant récupérer 100% d'or. Le consommation de cyanure était 1 kg/t.

Les essais à l'echelle du laboratoire pour les techniques de récupération a montré que le charbon activé et l'extraction par solvent rendent possible une récupération de plus de 97% d'or. La cémentation (Merrill-Crowe) et l'électro-extraction donnent résultats plus bas (85% et 71% respectivement).

On a estimé (± 30%) la coste d'une plante de 250,000 t/a être $1.4 milliones.

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1.0 INTRODUCTION

A number of potential gold ore bodies have been discovered in New Brunswick. With some exceptions including Cape Spencer, most of these ore bodies are refractory. The Cape Spencer ores are low grade but amenable to cyanidation. The refractory nature of some ores and the low grades of others has resulted in very few of the deposits being exploited. Only Nova Gold's Murray Brook Resources deposit near Bathurst is currently operating. Anaconda treated a gossan cap at the Caribou site near Bathurst and Gordex operated a heap leach plant in the Cape Spencer area until recently.

As part of their mandate to promote mineral exploitation and development, the Mineral Development Branch of the Department of Natural Resources and Energy has undertaken limited laboratory testwork to determine processing options for some of these orebodies (1, 2). This work was carried out under the Canada - New Brunswick Mineral Development Agreement.

This testwork showed that Cape Spencer ores were leachable with sodium cyanide but it also showed that there were high levels of other ions, in particular copper, which were dissolving and which could interfere with the recovery of the gold. Therefore, it was decided that larger scale testing of Cape Spencer ores should be undertaken to determine the effects of these interfering ions. This project was also funded by the Mineral Development Agreement.

This report presents the data generated in the course of this "Study of The Processing Options of Cape Spencer Ores". The project encompassed the following four tasks:

i to review and assess existing and developing technologies for the recovery of gold from pregnant heap leach solutions

ii to determine the best recovery technology based on laboratory tests and comparative economic assessments

iii to present recommendations for the design and operation of a pilot scale heap leach and recovery plant

iv to evaluate thiourea as an alternative leachant to sodium cyanide

The main stages in the project were a state-of-the-art literature review, a mineralogical examination, laboratory testwork, large scale leaching column testwork, and capital and operating cost determinations.

2.0 LITERATURE REVIEW

2.1 Heap Leaching Technology

Heap leaching to recover metals has a long history. For example, in mid-sixteenth century Hungary copper-bearing solutions were recycled through waste heaps (Hiskey,3) and in the mid-eighteenth century, along the banks of the Rio Tinto in Spain, heaps of oxide copper ores were leached with acid. Copper continues to be recovered from its oxides by sulphuric acid leaching; Chuquicamata, in Chile, has recently started a large scale copper leaching operation. Heap leaching for uranium has been practised for 30 years.

Application of heap leaching for the recovery of gold using cyanide was first proposed by the U.S. Bureau of Mines in 1967 (Thorsted, 4). The first commercial heap leach operation was operated by Carlin Gold Mining Company in Nevada in the late 1960's (Hiskey, 3). The first large scale operation was started in the early 1970's by Cortez Gold Mines (Thorsted, 4). Since this start approximately two decades ago heap leaching operations have been operated all over the world.

Heap leaching constitutes a very simple process. The ore is crushed and delivered to a specifically prepared pad where it is piled up to a given height (typically 3 to 10 meters (10-30 feet)). Alkaline cyanide solution is applied to the top surface and allowed to percolate down through the heap. The pregnant solution is collected at the bottom and sent to a gold recovery process. The barren solution is recirculated after the addition of make-up reagents.

Heap leaching has lower capital and operating cost and can be started up more quickly than conventional gold processing plants (Dorey et al, 5). This makes the technique suitable for treating small or low grade finely disseminated orebodies that would otherwise be uneconomic. Gold deposits with grades as low as 0.65 g/t (0.02 oz/ton) are currently being processed (Dorey et al, 5).

From its initial use in the western United States, this technology is now being profitably applied in Australia, Asia, Canada, Chile, Mexico, Peru and Central and South Africa (von Michaelis, 6). As these new areas emerge new operating constraints e.g. extremely high rainfall, very arid locations, hot and cold temperature extremes and high altitudes, are introduced. Another improvement in the application of the technology was the ability to handle clayey deposits or those requiring a fine grind through agglomeration. This improvement also enabled mill tailings to be heap leached.

For an ore to be amenable to heap leaching it must have certain characteristics:

- The precious metals must be recoverable in a suitable solvent.
- The host rock must be sufficiently permeable to allow the solvent to reach the precious metal.
- The ore must be free of "preg-robbing" constituents which can cause premature absorption or precipitation of the precious metal.
- The ore must be relatively free of cyanicides that consume cyanide or interfere with dissolution.
- The ore should be relatively free of fines or clays that impede percolation (agglomeration can treat this problem).
- The ore must be free of acid forming constituents that cause high cyanide or base consumption. (Heinen et al, 7).

Permeability of the ore and percolation through the heap are two very important parameters (Thorndycroft, 8). Permeability refers to the ability of the solvent to penetrate and contact the ore particles below the exposed surface. This can be achieved by absorption along grain boundaries or cracks. With increasing imperviousness of the host rock finer crushing and/or grinding and longer leaching times are required to recover the precious metal(s).

Percolation refers to the gravity flow of the solution through the heap. Percolation can be affected by many variables including size distribution and heap construction. Excess

fines either in the ore or produced during heap construction can block the solution flow and in the worst case can result in ponding on the heap surface and then run off down the sides of the heap. Further, fines tend to migrate to the bottom of the heap and form an impermeable barrier which can eventually result in solution coming out through the sides of the heap or in mobilizing the bottom of the heap at which time the whole heap could collapse. Channelling can be caused by improper heap construction and is the consequence of the leach solution following selected, confined paths through the heap. This results in minimal contact with the contained precious metals which can result in very poor recoveries.

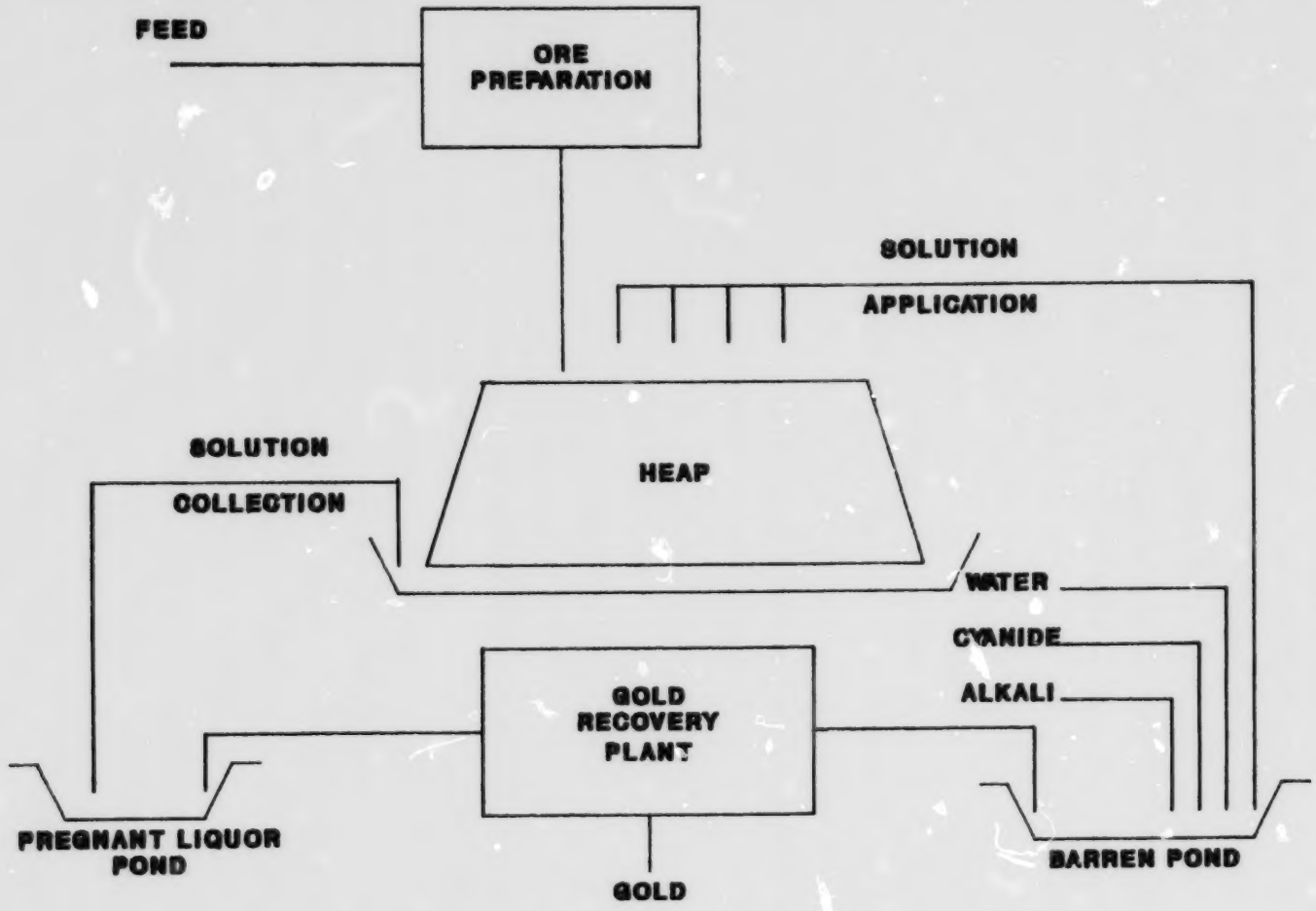
A heap leach operation consists of the following basic unit components (Dorey et al, 5).

- Mine or ore source
- Ore preparation including materials handling and reagent preparation
- Pad and heap
- Solution application and collection systems
- Pregnant solution storage
- Precious metal recovery system
- Barren solution storage

This is illustrated in Figure 1.

Ore preparation includes all activities related to transporting the ore from its source; any crushing or grinding that is required; agglomeration of the fines if required; and transporting the prepared ore to the leaching site. The metallurgy of the ore dictates the pretreatment methods to be used and in turn the pretreatment technique may dictate the heap operation and construction. The objective of ore pretreatment is to obtain a material that is sized so that both the recovery is maximised and the stability of the heap is optimised.

FIGURE 1
SCHEMATIC OF HEAP LEACH OPERATION



One pretreatment technique which may be pertinent to Cape Spencer ores because of the fairly high fines content is agglomeration. This is basically a pelletizing process where the fines are mixed with cement and water or a concentrated cyanide solution. The latter is better to use because gold recovery seems to be enhanced when using cyanide in the pellet make-up. The optimum moisture content is approximately 13%.

Factors which affect the heap and pad include ore preparation, leachability, site topography, site geotechnical and geohydrological characteristics and site climate (Dorey et al, 5). Two types of pads currently used are, reusable, where the ore is stacked and removed after leaching and permanent, where the ore is left after stacking. Asphalt is a common material used in reusable pads while clay and/or plastic are used for permanent heaps.

Ore placement on the heaps can be done by dumping from trucks, placing by front end loaders or conveyor stacking. The main constraints are limiting layering, compaction and segregation in the heap and preventing damage to the liners.

2.2 Leachants

Sodium cyanide (NaCN) is the leachant predominantly used to dissolve and recover gold. Calcium cyanide ($Ca(CN)_2$) is used to a lesser extent. NaCN is approximately 98% pure while commercial $Ca(CN)_2$ is approximately 49% pure. The use of $Ca(CN)_2$ is based on economics rather than performance although there are examples where one has performed better than the other.

Other gold leachants include thiourea; thiosulphate; brines; and halides, particularly bromine.

2.2.1 Cyanide

The basic idea for cyanidation was first patented in 1846 by the German chemist L. Elsner. Forty years later the importance of the discovery was recognized by J.S. MacArthur, a metallurgical chemist, and R.W. and E. Forrest, medical doctors, all of Glasgow, Scotland. They acquired patents in Great Britain in 1887 and in the United States of American in 1889. The basis of the McArthur - Forrest process was the use of a weak (0.05%) cyanide solution instead of the 1.0% cyanide solution.

Traditionally, the overall equation for the dissolution of gold is accepted as being:

$$4\ Au + 8\ CN^- + O_2 + 2H_2O = 4\ Au(CN)^-_2 + 4OH^- \qquad (1)$$

More recent research has shown that most of the gold dissolves by the reaction:

$$2\ Au + 4\ CN^- + O_2 + 2H_2O = 2\ Au\ (CN)^-_2 + H_2O_2 + 2OH^- \qquad (2)$$

with the rest dissolving via Elsner's equation - Equation 1 (Adamson, (15)).

In a simple ore the gold will dissolve readily. The only requirements are that the gold be free and clean, that the cyanide solution contains no detrimental impurities and that sufficient oxygen is present during the reaction.

The rate of dissolution of gold in cyanide is affected by cyanide concentration, particle size, oxygen, alkalinity and temperature. J.S. Maclaurin (9) showed that maximum dissolution occurred at 0.25% NaCN, Christy (10) found that solutions weaker than 0.001% KCN do not dissolve gold, Julian and Smart (11) found that the rate of dissolution of gold increased as the strength increased up to 0.10% KCN, White (12) showed that maximum dissolution occurred at 0.027% KCN or 0.02%

NaCN when the cyanide solution is saturated with oxygen and Barsky, Swainson and Hedley (13) found that the most rapid dissolution was at 0.05% NaCN. These various findings probably differ due to variations in experimental technique.

If the ratio of cyanide solution volume to exposed gold surface is large and sufficient agitation is available to remove reaction products from the surface as they form, the rate limiting factor is the available oxygen content. At sea level and using air, the maximum oxygen content of the cyanide solution is 8 mg/litre. According to Equation 1, the maximum required amount of NaCN is 392 parts for every 32 parts of oxygen or 98 parts of NaCN for every 8 parts of oxygen (Hedley and Tabachnik, 14). Therefore, under ideal conditions the maximum rate of dissolution will occur when the cyanide solution strength is 0.0098%. In practice the cyanide solution strength is less than 0.05% and usually is 0.02 or 0.03%.

Particle size is a factor in that larger particles require more time to dissolve in the cyanide solution. Barsky et al (13) found that under ideal conditions the maximum rate of dissolution of gold was 3.25 mg per cm^2 per hour. This equates to a removal rate of 1.68 μm from each side of a flat particle. Therefore a particle 44 μm thick (325 mesh) would dissolve in 13 hours while a 149 μm (100 mesh) particle would take 44 hours.

Oxygen or an oxidizing agent is essential for leaching to occur. Barsky, Swainson and Hedley (13) showed that the rate of dissolution of gold in cyanide solutions is directly proportional to the oxygen content up to a maximum level beyond which oxygen ceases to be a limiting factor for dissolution.

The ore may contain metals or other components which are oxygen consumers. If present in large enough amounts these components can have a deleterious effect on the cyanidation reaction. Loss of oxygen from the percolating solution was one of the factors which affected heap height though experience has shown that heaps up to 15 metres (50 feet) can be leached (Milligan and Muhtadi, 17).

Oxidising agents such as sodium peroxide, potassium permanganate, bromine and chorine have been used in the past with some success. Their cost and handling problems have eliminated their use with cyanide. Bromine and chlorine may have a role as alternatives to cyanide in leaching of gold.

The dissolution of gold is sensitive to the pH or alkalinity of the solution. The reasons for operating under basic conditions include:

- prevention of loss of cyanide by hydrolysis
- prevention of cyanide loss by the action of carbon dioxide in the air
- water treatment to decompose bicarbonates or neutralize acidic compounds such as ferrous salts, ferric salts and magnesium sulphate in the process water
- neutralization of acidic components of the ore or acidic compounds formed by the decomposition of minerals in the cyanide solution
- improvement in the extraction of ores containing for example ruby silver or tellurides which decompose more readily at high pHs.

Lime is the pH modifier which is most often used. Sodium carbonate and sodium hydroxide can also be used. Lime can help in clarifying the pregnant liquor. The optimum dissolution rate is found frequently to occur in the vicinity of pH 10.5 and the dissolution rate of gold is inhibited above pH 12 and below pH 10 where there is a danger of hydrolysis occurring. Above pH 11, the degeneration of pyrrhotite is usually accelerated, however, for some pyrrhotite ores pH 12 or higher gives the best results.

pH control in a heap can be accomplished by adding pH control reagents to the heap ore material during construction as well as maintaining the pH of the circulating solution.

Two opposing factors affect the rate of dissolution when heat is applied. With increasing heat, the activity of the solution will increase and the rate of

dissolution will increase. At the same time, the oxygen content of the solution will decrease. The optimum leaching temperature is approximately 85°C. Conversely, with decreasing heat the activity and thus the rate of dissolution decreases and increased oxygen as was already been explained does not mitigate the decreased dissolution rate. The leaching rate for a heap in the winter may be 70% of the summer rate.

While some ores are relatively simple, many ores have constituents which will react to some degree with the cyanide solution or react with other components to the detriment of gold dissolution or recovery. Pyrite, pyrrhotite, marcasite,chalcopyrite, arsenopyrite, enargite, sphalerite, galena, tetrahedrite, tennantite and stibnite are commonly associated with gold and are soluble in cyanide solutions.

Solubilities of copper and zinc minerals in cyanide solution are shown in Tables 1 and 2.

Metals in solution form complexes with the cyanide which reduces the free cyanide content. Some complexed cyanide may be recaptured by gold. Three factors control this. The first is the relative strength of the gold-cyanide complex versus the strength of the metal-cyanide complex. The second is the rate at which metal-cyanide complexes are dissociated i.e. the rate must be high enough to effect the transfer of cyanide. Thirdly, a second metal may be able to remove cyanide from a metal-cyanide complex if the complex to the second metal ratio is high.

Copper forms a very strong complex with cyanide. If sufficient copper is in solution it can remove all of the free cyanide. Options to solve this problem include adding sufficient cyanide to complex all the copper and still have enough cyanide to provide for the gold-cyanide complex and the required free cyanide; and removal of the copper from the solution. The zinc-cyanide complex will decompose to provide cyanide for gold, cobalt (3+), mercury, nickel, copper and silver complexes. The iron-cyanide complex does not dissociate sufficiently rapidly, so any cyanide complexed with iron is effectively lost.

TABLE 1

SOLUBILITY OF COPPER MINERALS IN 0.099 PERCENT NaCN SOLUTIONS

Mineral		Percent Total Copper Dissolved At 23°C	At 45°C
Azurite	$2CUCO_3 \cdot Cu(OH)_2$	94.5	100.0
Malachite	$CuCO_3 \cdot Cu(OH)_2$	90.2	100.0
Cuprite	Cu_2O	85.5	100.0
Chrysocolla	$CuSiO_3$	11.8	100.0
Chalcocite	Cu_2S	90.2	15.7
Chalcopyrite	$CuFeS_2$	5.6	8.2
Bornite	$FeS \cdot 2Cu_2S \cdot CuS$	70.0	100.0
Enargite	$3\ CuS \cdot As_2S_5$	65.8	75.1
Tetrahedrite	$4\ Cu_2S \cdot Sb_2S_3$	21.9	43.7
Metallic Copper	Cu	90.0	100.0

TABLE 2

SOLUBILITY OF ZINC MINERALS IN CYANIDE SOLUTION

Mineral	Zinc Present		Percent [1] Extraction of Zinc
	Heads	Tailings	
Sphalerite, ZnS	1.36	1.11	18.4
Willemite, Zn_2SiO_4	1.22	1.06	13.1
Hydrozincite, 3 $ZnCO_3$, 2 H_2O*	1.36	0.87	35.1
Calamine, (Hemimorphite), $H_2Zn_2SiO_2$	1.19	1.03	13.4
Franklinite (Fe, Mn, Zn)O·(Fe, Mn)$_2O_3$	1.19	0.95	20.2
Zincite, ZnO	1.22	0.79	35.2
Smithsonite, $ZnCO_3$	1.22	0.73	40.2

* Formula usually given for hydrozincite is 2 $ZnCO_3$·3 $Zn(OH)_2$.

(1) The zinc minerals were mixed with finely ground quartz so that each mixture contained approximately 1.25% Zn. The mixtures were agitated in 0.197% NaCN solution for 24 hours at 45°C.

2.2.2 Thiourea

The dominant leachant for gold since its introduction approximately 100 years ago has been cyanide. Other reagents have been investigated and occasionally plant tested. One that has received much attention recently has been thiourea. Thiourea is not detrimental to the environment, is non-toxic to humans and dissolves gold faster than cyanide solution can.

Thiourea (H_2NCSNH_2) is an organic compound with an aqueous form which is stable in acidic solutions up to pH 4.3. It extracts gold by forming a stable cationic complex. It is believed that formamidine disulphide is the complexing agent and this is formed at pH 1 and at a given solution oxidation potential. An oxidising agent such as ferric sulphate is used to achieve and maintain the oxidation potential.

The dissolution of gold by thiourea is expressed by

$$Au + (NH_2)_2\,CS = Au\,(NH_2)_2\,CS^{+} + e^{-} \qquad (3)$$

Thiourea undergoes various stages of oxidation. The first, to formamidine disulphide, occurs at anodic potentials greater than 0.30V, is fairly rapid, and is reversible. Succeeding reactions are much slower and form oxidation products which include elemental sulphur and sulphate ions. The presence of sulphate ions slows the reaction between ferric ions and thiourea because of the stabilizing effect of the $[Fe\,(SO_4)\,(NH_2)_2\,CS]^{+}$ complex. These later oxidation stages are irreversible and thus can lead to significant thiourea losses. There is, therefore, a narrow range of oxidation potentials in which both satisfactory gold leaching occurs and reagent consumption is minimised (Filener et al, 22).

The overall rate of dissolution of gold is generally controlled by the diffusion of either reactants to the gold surface or products from the surface.

The dissolution of gold in acidic thiourea solutions was first reported in 1941 by Plaksin and Kozhukhova (19). Since then the use of thiourea has been examined extensively. Tatam (20) determined that the optimum concentration for thiourea was between 90 and 100 g/L. Studies of the influence of Fe^{3+}, Cu^{2+}, Zn^{2+} and Pb^{2+} have shown that at low concentrations Fe^{3+} enhances the rate of dissolution but at high concentrations it is deleterious. In all cases the rate of dissolution decreases as the metal ion concentration increases, and lead produces the most deleterious effects.

Lodeischileov et al (21) in tests using a thiourea - sulphuric acid - ferric ion system found that optimum leaching conditions corresponded to 4 g/L thiourea, 5 g/L H_2SO_4 and 3 g/L $Fe_2(SO_4)_3$. Their gold recoveries after eight hours were 92-98% depending on ore type. Preliminary acid washing was required to maximise recoveries.

The dissolution rate of gold in thiourea solutions is faster than in cyanide solutions; complete dissolution of the accessible gold usually occurs within a few hours.

The orebody constituents will affect thiourea consumption and its ability to recover gold. The constituents must be consistent with achieving the required potential range. Because the gold-thiourea complex is cationic it is more susceptible to absorption on clays than anionic species. Although absorption is minimised under acidic conditions, the effect can still be significantly detrimental. Some constituents of an ore may absorb thiourea or formamidine disulphide which increases reagent consumption. Pyper and Hendrix (23) found that copper and manganese ions produced drastic increases in thiourea consumption.

Increasing acidity causes increased thiourea consumption (23, 24) probably by catalyzing the hydrolysis reaction to form urea and hydrogen sulphide.

Control of the redox potential is necessary to minimise thiourea consumption. Bilston et al (24) found 150 mv (versus SCE) to be adequate, while Groenewald (25) suggested 160 mv for the dissolution of metallic gold and Pyper and Hendrix (23) found 150-160 mv was sufficient to obtain 80-85% extraction from Carlin type carbonaceous ores, with no improvement at higher potentials.

Gold extraction increases with increased thiourea concentration, (24, 25, 23, 19), however, thiourea consumption also increases with increased concentration. The rate of thiourea consumption is initially rapid but slows down after approximately two hours.

Although thiourea is relatively selective for gold over other metals, small amounts of copper, iron, lead and zinc will dissolve. Bilston et al (24) working with an oxidised gold ore from Nevoria, Australia found that metal concentrations in solution were about 3 ppm Cu, 2700 ppm Fe, 2 ppm Pb and 12 ppm Zn. Maximum Cu and Zn extraction occurred at pH 1.5 while all metal extractions declined with increasing pH.

2.2.3 Thiocyanate

White (26) demonstrated in 1905 that thiocyanate could dissolve gold. Nothing further was done with this leachant until Fleming (27) in 1986 published his work using a sulphate-thiocyanate system to leach South African ores.

Thiocyanate is environmentally more acceptable than cyanide. Also, soluble oxidising agents can be used since the leaching occurs in acidic (pH <3.0) medium. Na^+, K^+ or NH_4^+ thiocyanate salts can be used since they have high solubility in aqueous solutions.

The thiocyanate ion $(SCN)^-$ is an ambidentate ligand which can form two series of covalent complexes, thiocyanates or S- thiocyanates and isothiocyanates or N- thiocyantes. Gold and silver usually complex as S- thiocyanates.

At potentials of 600-700 mV (versus SHE), at concentrations of 29 to 0.6 g/L and at pH 1 to 3, thiocyanate can leach gold. The gold is expected to dissolve as the complexes $Au(SCN)_2^-$ and $Au(SCN)_4^-$. Oxidants such as ferric ions, hydrogen peroxide or Caro's acid can be used to enhance the dissolution.

2.2.4 Chlorine

The earliest use of chlorine in metallurgy appears to predate 1540 because a treatise, de la Pyrotechnica, published in 1540 in Venice asserted that a wet salt process was used to recover silver. The earliest recorded use of chlorine was in 1557 by Bartholme Medina. With various modifications the leaching of silver ores with chlorine continued until 1893 when the U.S. Government demonetized silver.

The use of chlorine for recovery gold was initiated in 1851. The gold ore was dead roasted, then chlorine gas was applied in vats. The gold was recovered by subsequent leaching with water. Gold losses associated with this process were quite high and it was some time before it was discovered that gold chloride was volatilizing.

With this discovery came a long series of experiments and techniques of which the best known is the Pohle-Croasdale procedure. Although the use of chlorine and chlorides was not economic compared to cyanide experimental work has continued.

The use of chlorine gas required using elevated temperatures at some point in the process. This is impractical for a heap leaching operation and the details of the process will not be covered. A combination of chlorine gas and brine can be used as a leachant for gold.

Chlorine forms a weak complex with gold. Very oxidizing conditions are required for gold dissolution otherwise metallic gold will reprecipitate. The dissolution of gold by chlorine is expressed by

$$3/2\ Cl_2 + Au + Cl^- \rightarrow AuCl_4^- \qquad (4)$$

A brine solution is used to provide the chloride ions and chlorine in the oxidant.

The rate of dissolution of gold is very fast under acidic conditions: gold can be extracted in one hour. A pH level of 3.0-3.5 is required for optimum leaching. Above pH 4.0 the leaching rate slows and at less than pH 3.0 the chlorine gas forms hypochlorous acid. Operating outside this pH range requires the use of a pressurized vessel.

The use of chlorine is affected by the sulphide content of the ore. Pyrite and other sulphides have a dissolution rate similar to gold and if not completely dissolved they can reduce the soluble gold. Ores which contain more than 0.5% S are considered unsuitable for leaching by chlorine.

Other potential drawbacks include the extremely corrosive nature of acidic chlorine solutions. This means that materials of construction must be chosen carefully.

2.2.5 Bromine

Bromine chemistry is quite similar to chlorine chemistry. There are, however, some important differences between the two elements; chlorine is a gas while bromine is a dense, fuming, red liquid. The first patent for the use of bromine in gold extraction was issued in 1881 and by 1895 bromine was considered more cost effective than chlorine even though on a pound for pound basis it was more expensive. With the advent of cyanidation, the use of halides was virtually eliminated.

Bromine acts by oxidising the gold to form a water soluble salt. The oxidation occurs in two stages. Three parts of hypobromous acid react with two parts of gold to form gold tribromide which reacts with bromite ions to form tetrabromo gold ions.

$$2\,Au + 3\,HOBr + 3Br^- \rightarrow 2\,AuBr_3 + 3OH^- \quad (5)$$

$$AuBr_3 + Br^- \rightarrow AuBr_4^- \quad (6)$$

Several bromine salts are available which could provide a more convenient source of bromine rather than feeding the liquid itself. Brominated hydantoins and salts e.g. dibromodimethylhydantoin hydrolize to give free bromine and dialkylhydantoin. The bromine reacts with the water to form hypobromous acid.

2.3 Recovery Processes

The most important gold recovery processes are zinc precipitation (Merrill-Crowe), carbon adsorption, ion exchange, solvent extraction and electrowinning.

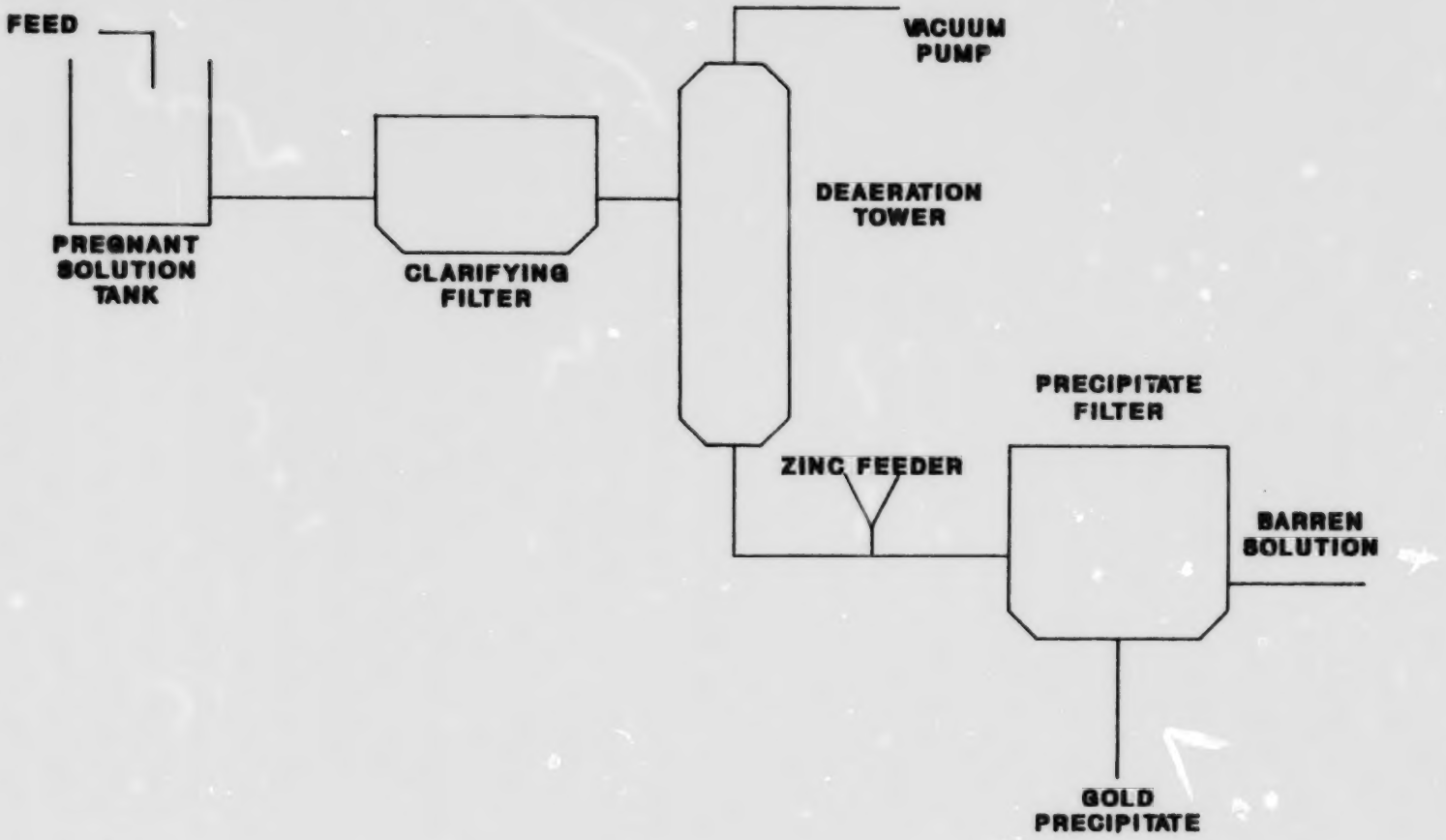
2.3.1 Zinc Precipitation

Zinc precipitation with modifications has been used since 1888 to recover gold and silver from cyanide solutions. The process is particularly applicable to solutions where the silver to gold ratio is high. Nova Gold's Murray Brook operation in Northern New Brunswick uses zinc precipitation for this reason. The gold cements onto the metallic zinc surface, as shown by the following reaction:

$$2Au(CN)_2^- + Zn = 2Au + Zn(CN)_4^{2+} \quad (7)$$

A conceptual flowsheet for the Merrill-Crowe process is given in Figure 2. The pregnant liquor is stored in a tank prior to feeding to a clarifying filter on a vee cone thickener or hydrocyclone. The solution must be solids free prior to adding the zinc dust. The clarified solution is drawn by vacuum into the Crowe tower where deoxygenation to less than 0.5 mg/L occurs. Zinc dust is added to the pregnant solution usually at a rate far in excess of stoichiometry. Typical consumptions are 1-2g Zn per g of Au. Lead nitrate or acetate at a dosage of approximately 10% of the zinc addition may be added with the zinc dust. This establishes a powerful

FIGURE 2
SCHEMATIC OF MERRILL-CROWE PROCESS



galvanic couple which enables gold to be recovered from solutions low in free cyanide concentration. The solution pH needs to be above 10 in order for precipitation of precious metals to occur. This pH will minimize calcium precipitation.

The slurry of precious metal precipitate and unreacted zinc is pumped to a precoat pressure filter. Since the precipitation kinetics are fast no mixers or tanks are required between zinc addition and the pressure filter. Zinc dust can be added with the precoat material (usually diatomaceous earth) as a safety measure. A typical precipitate analyses between 10 and 40% Au. The grade can be increased by washing with pregnant solution or by dissolving excess zinc in sulphuric acid after removal of the precipitate from the filter press.

Aluminum can be used instead of zinc, however, the formation of calcium aluminate can result in filters becoming plugged. Also, aluminum can give problems in the further smelting of the gold.

Overall operation of the zinc precipitation process is affected by the various ions that solubilize with the precious metals. Sulphide ions at concentrations of 10-15 ppm will inhibit gold cementation probably by reacting with the zinc to form a surface layer of zinc sulphide which stops the zinc from contacting the gold. Copper in solution can cause high zinc consumption and will affect the efficiency of gold precipitation. Maintaining sufficiently high free cyanide levels will eliminate this problem by keeping the copper tied up in a copper-cyanide complex. Lead in concentrations greater than 20 ppm may result in higher zinc consumption.

Arsenic and antimony can have a drastic effect on the kinetics of precipitation. Levels of 1 ppm in the pregnant solution can reduce the reaction rate by 20%. Precipitation of the arsenic and antimony by sodium sulphate can alleviate this problem; however, silver will be lost as well.

2.3.2 Carbon Adsorption

Carbon adsorption works by adsorbing the gold in the pregnant solution onto activated carbon. A conceptual flowsheet is presented in Figure 3.

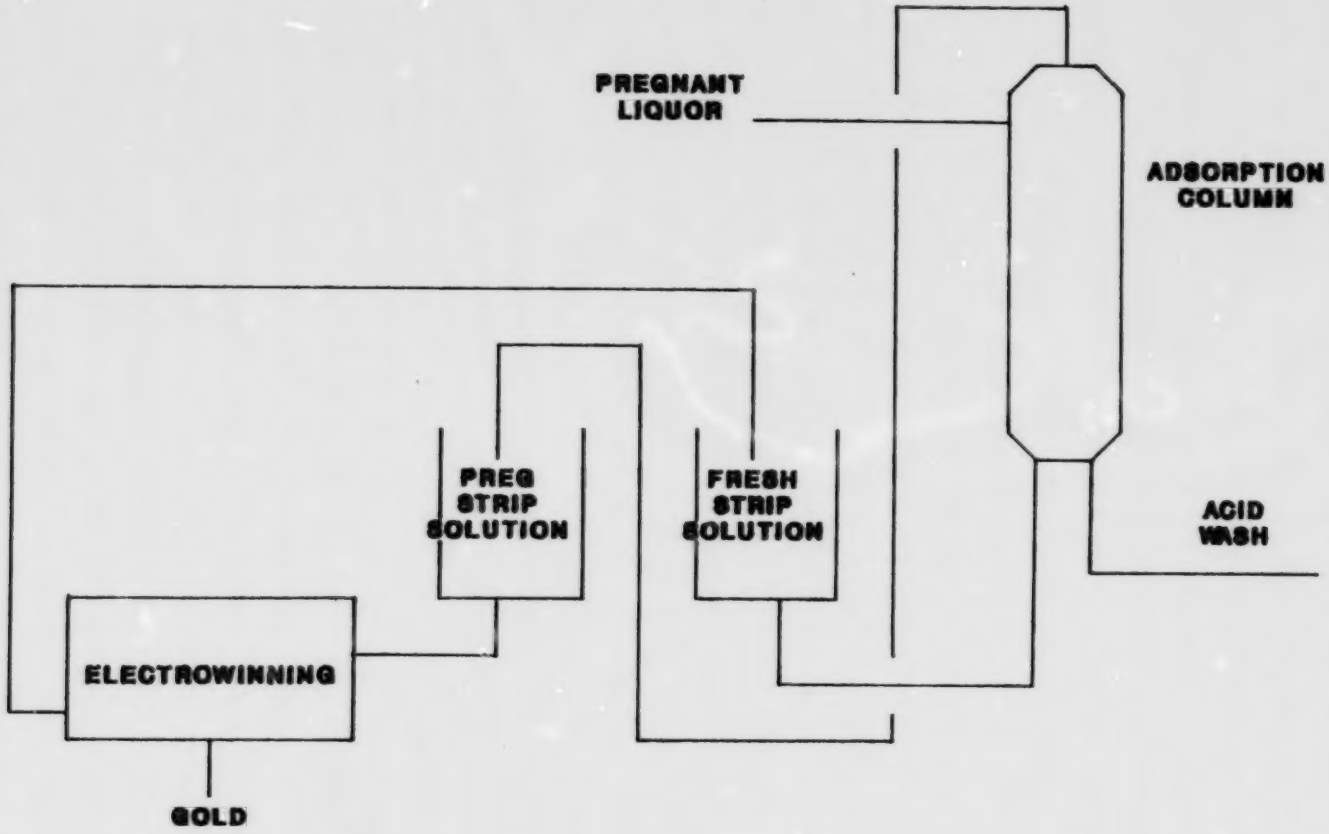
The pregnant solution is passed upwards through columns containing the activated carbon. Typical flowrates of 0.01 to 0.016 $m^3/sec/m^2$ (15 to 25 gpm/ft^2) are used to fluidize the carbon bed. The carbon particles are usually 500 μm to 1000 μm in size. The carbon is advanced countercurrently at a rate sufficient to achieve the desired gold loadings. A carbon loading of 6220 g/t (200 oz/ton) can be achieved. Excess loading results in gold losses to the barren solution.

Recovery of the gold from the loaded carbon can be achieved by a number of methods. The most common are:

a) **Zadra**: the gold is stripped using a solution containing 1% NaOH and 0.1% NaCN. The process is carried out at atmospheric pressure and at 85°C-95°C. Elution time can range from 24 to 60 hours depending on solution chemistry and mode of operation. The process is simple and has low capital and operating costs which makes it suitable for small scale operations.

b) **High Pressure Stripping**: this is basically a modification of the Zadra process. The NaCN is optional and the stripping is carried out at 160°C and 350 KPa (50 psi) pressure for 2 to 6 hours. Advantages of the process include lower reagent consumption, smaller carbon inventory and a smaller stripping section. Disadvantages are that more expensive equipment is required to handle the higher pressures and temperatures and that the effluent solutions have to be cooled before reducing the pressure.

FIGURE 3
SCHEMATIC OF CARBON/RESIN ADSORPTION



c) **Anglo American Process:** the loaded carbon is preconditioned for one half to one hour with a half bed volume of a solution containing 5% NaOH and 1% NaCN. Stripping is carried out with five bed volumes of hot water at 110°C at a rate of three bed volumes per hour. The operating pressure is 50 to 100 KPa. Total processing time from start to finish is nine hours. This process offers the same advantages as high pressure stripping and has the same disadvantages with the addition of having multiple streams which makes the circuit more complex.

d) **Alcohol Stripping:** this is a modification of the Zadra process. The strip solution contains 20% alcohol in addition to the 1% NaOH and 0.1% NaCN. The stripping takes place at 80°C. The alcohol reduces the stripping cycle time to 5 to 6 hours. Advantages are reduced cycle time and thus a smaller stripping section and possibly less frequent carbon regeneration. Disadvantages are a high fire risk and higher operating costs because of alcohol loss due to volatilization. Various safety features and a vapour recovery system are essential.

The gold can be recovered from the pregnant strip solution by electrowinning or by zinc precipitation. Zinc precipitation is the more complicated procedure of the two.

Electrowinning is the most commonly used recovery technique. Numerous cell designs have been proposed though most use steel wool as the cathode. Variables include cell shape, cylindrical or rectangular; anode sheets; electrolyte flow patterns; separation of anode and cathode compartments by permeable membranes; and packing density of the steel wool in the cathodes. The National Institute for Metallurgy in Johannesburg, South Africa have worked on a cell design which uses a sandwich consisting of graphite granules for the cathode separated from the two anodes by permeable cation exchange membranes. Gold deposits on the carbon

granules and is recovered into an aqueous solution of potassium hydroxide and potassium cyanide by reversing the cell polarity. The solution is then circulated through a plating bath where the gold is deposited on titanium starter sheets. The gold foil can be peeled off the starter sheets and melted to produce gold bars with greater than 99% purity. With this cell, it is not necessary to remove the cathode to recover the gold.

Regeneration of the carbon is an important part of the carbon-in-column process. Various inorganic and organic species get adsorbed along with the gold and eventually these poison the carbon. Regeneration is accomplished in two stages. Acid washing is used to remove calcium carbonate and most base metal complexes. Acid washing can be done before elution, after elution and before thermal reactivation, or after reactivation. Care must be taken when washing prior to reactivation because HCN gas can be generated. Washing after reactivation is not recommended because some contaminants interfere with thermal reactivation. It is possible that acid washing prior to elution can give better gold elution. In all cases, the carbon must be neutralized and water washed after the acid wash and before moving to the next stage in the process.

Thermal reactivation involves heating the carbon to approximately 650°C for up to 30 minutes in the absence of air. A rotary kiln is the most commonly used reaction vessel. After reactivation the carbon is cooled with water or air and screened to remove fines. Air cooling has been shown to give higher carbon activity.

The type of carbon used is very important to the economics of the process. Activated carbon prepared from coconut shells is preferred because it has a large surface area (>1000 m^2/g), narrow range of pore sizes (10-20 A) and good strength so that the carbon does not abrade easily thereby losing gold.

An advantage of carbon-in-column over zinc precipitation is that it can handle solutions containing suspended solids.

2.3.3 Ion Exchange

Ion exchange uses resin-in-columns in a manner similar to carbon-in-column (see Figure 3). The loading section consists of a series of columns containing the resins. Resin beads are generally smaller and less dense than carbon granules and require the application of lower flowrates to avoid carry-over. Typical flows are 0.004 $m^3/sec/m^2$ (6 gpm/ft^2). Therefore, reactor sizes are relatively larger.

Weak or strong base resins can be used for adsorption of the gold from the pregnant liquor. The loading reactions are as follows:

Weak base

$$-NR_2 + H^+ + x^- = -NR_2\,H^+\,x^- \quad (8)$$

$$-NR_2H^+x^- + Au(CN)_2^- = -NR_2H^+\,Au(CN)_2^- + x^- \quad (9)$$

Strong base

$$-NR_3^+x^- + Au(CN)_2^- = -NR_3^-\,Au(CN)_2^- + x^- \quad (10)$$

As these equations show, the weak base resins require protonation in order to extract anions. This occurs at pH values less than 10, therefore, the pregnant solution may require some pH adjustment in order to obtain acceptable loadings. Maximum extraction with weak base resins occurs at about pH 6 while extraction by strong base resins is independent of pH. Extractions by weak base resins at typical pH values for a cyanide system are approximately 20%.

The elution process is dependent on the type of resin used. Dilute caustic solution can be used to recover the gold from a weak base resin. Strong base resins are harder to elute. Concentrated $Zn(CN)_4^{2-}$ has been shown to be effective (Fleming and Nicol, 23) but the resin then needs regenerating with acid which destroys the $Zn(CN)_4^{2-}$ and produces HCN gas.

Ion exchange resins can be used to recover gold from a thiourea leach solution since the gold-thiourea complex is readily adsorbed on a cationic resin. Bjerre and Sorensen (28) on their work with a strong-acid cation exchanger found that the gold can be eluted by a ligand exchange process rather than an ion exchange process. Typical leach solutions contain other metal ions, particularly iron, which are adsorbed along with the gold. These metals do not form as strong an attachment as the gold and can be selectively removed, prior to eluting the gold, by washing the resin with nitric acid and hydrogen peroxide.

The gold can be removed by washing with $(NH_4)_2\ S_2O_3$. The $S_2O_3^-$ ion replaces the thiourea ion in the gold complex and forms $Au\ S_2O_3^-$ which, since the ion exchanger has no affinity for anions, leaves the column. The desorption of gold requires a pH of 5 to 6.5.

2.3.4 Solvent Extraction

The use of solvent extraction to recover gold from gold leach liquors is relatively new and is not in commercial use. Solvent extraction is used by precious metal refiners to recover precious metals including gold from various feed sources such as copper anode slimes and spent catalysts. The gold is usually present in the leach solutions as the chloroanion. Therefore, this process would be more readily applicable to pregnant solutions which used chlorine/chlorides as the leachant.

Most of the current commercial processes use aliphatic amines as the extractants. These amines can be oxidized to amine oxides which can degrade to products such as N, N- dialkyl -hydroxylamines. These products combined with hydrolysis products can interfere with the recovery process. Alternatives which have been proposed include hetrocyclic N- oxides such as 2- nonylpyridine -N- oxide (Hudson and Glaves, 29), 1, 3, 4- thiadiazole -2- nonylmercapto -5- thiol (Hudson and Shepherd, 30) and 2- mercaptobenzothiazole (Shepherd, 31 and Diamantos, 32).

The recovery of gold from cyanide solutions can be effected by weak base amines whose basicity has been increased by an organic phosphorous modifier such as tributylophosphate (TBP). Gold loading is accomplished between pH 9 and 10. The optimum pH depends on amine type and concentration, modifier type and concentration and possibly the carrier.

The loading reaction for secondary amine is as follows:

$$R_2NH + H^+ + Au(CN)^-_2 = [R_2NH_2^+]\,[Au(CN)^-_2] \qquad (11)$$

where R = C_{10} - C_{12} alkyl group

Stripping can be accomplished by raising the pH to 12 to 13 or with a 0.1 to 0.5% caustic solution.

Reaction rates are relatively rapid, therefore, long contact times are not required. This will affect reactor size. Selectively for gold over other metal-cyanide complexes viz $Fe(CN)^{3-}_6$ and $Cu(CN)^{2-}_3$ is good. Application of the process is still basically at the bench scale, therefore, reactor design is at a very elementary stage. Basic requirements of the process are high aqueous to organic flowrates and very low organic losses.

The process can tolerate a low level of suspended solids. Too high a level can result in excessive solvent losses (by adsorption) and the formation of interfacial crud which presents many problems in terms of recovery and operation.

2.3.5 Direct Electrowinning

Electrowinning has been used for many years to recover gold from solutions containing high concentrations of gold e.g. from strip solutions from carbon adsorption columns or resin ion-exchange columns. At concentrations of 50 to 2000 ppm the gold can be electro-deposited onto stainless steel wool cathodes.

Recent developments have focused on applying this technology to solutions containing 1 to 10 ppm gold. The key to this technique appears to be the use of "extended surface area" electrodes. Two forms have been tested: a flow through supported fibre electrode and a packed bed or particulate carbon electrode. The use of such electrodes increases the mass transfer limiting currents by reducing the boundary layer because of increased flow velocities and by increasing the specific surface area.

The gold can be recovered from the electrodes by a number of techniques depending on the type of electrode. For the particulate, packed-bed electrode, the carbon can be burnt in a melting furnace or the carbon can be treated chemically as for the carbon-in-column recovery process. For the fibrous electrode, electrolytic or chemical methods can be used to recover the gold.

Any silver, copper and mercury will tend to co-deposit with the gold. No information has been reported for zinc. The relevant electrochemical reactions are:

cathode:	$Au(CN)^-_2 + e \rightarrow Au + 2CN^-$	$E^\circ = -0.64V$	(12)
	$Ag(CN)^-_2 + e \rightarrow Ag + 2CN^-$	$E^\circ = -0.31V$	(13)
	$Cu(CN)^{3-}_4 + e \rightarrow Cu + 4CN^-$	$E^\circ = -0.75V$	(14)
	$Cu(CN)^-_2 + e \rightarrow Cu + 2CN^-$	$E^\circ = -0.43V$	(15)
	$O_2 + 2H_2O + 4e \rightarrow 4OH^-$	$E^\circ = -0.40V$	(16)
anode:	$CN^- + 2OH^- \rightarrow CNO^- + H_2O + 2e$	$E^\circ = 0.97V$	(17)

The cyanate ion reacts further to yield CO_2 and NH_3.

As the anodic reaction equation shows, cyanide can be destroyed. This can be avoided by using a diaphragm electrolysis cell which isolates the anode from the cyanide containing solution.

Prior to electrolysis, the pregnant solution has to be filtered to remove suspended solids. The barren solution is recycled to the heaps after reagent make-up.

A high surface area carbon-filter electrolysis unit has been pilot tested in Nevada. This unit manufactured by HSA Reactors Limited (1982) of Toronto reportedly can achieve acceptable metal recoveries from solutions with concentrations of 0.32 ppm and at a flowrate of 0.025 m^3/sec (400 gpm).

Mastri et al were able to obtain satisfactory results with flows of 0.011 m^3/sec (180 gpm) using 10 μm diameter carbon-graphite fibre threads with a specific surface of 0.2 m^2/g.

The U.S. Bureau of Mines have researched a bench scale unit which operates at ambient temperatures at 3V. Gold recoveries from a solution containing 7 ppm Au were up to 97% at an estimated cost of U.S $0.03/g. Above 3.5V the solution breaks down to form hydrogen and ammonia.

2.4 Overview

An overview of some of the technological features of the various processes is as follows:

Merrill-Crowe Process

Fast Reaction Kinetics
Low Selectivity
Requires Low Total Suspended Solids (TSS)
Requires Low Oxygen Concentration in Pregnant Leach Liquor
Easy to Operate
Good Accounting, Low Inventory
Can produce a product ready for melting

Carbon-in-Column

Slow Adsorption/Desorption Kinetics
Fair Selectivity
Low Sensitivity to TSS
Good Recoveries
Carbon requires moving and regenerating
Poor Accounting, High Inventory
Requires a second process to recover the gold from the carbon

Resin-in-Column

Intermediate Reaction Rates
Fair Selectivity
Insensitive to TSS
Flowrates are lower than for carbon-in-column, therefore, much larger equipment required
Requires a second process to recover the gold from the resin

Solvent Extraction

Fast Reaction Kinetics
Good Selectivity
Sensitive to TSS
Organic losses can be a problem and must be minimized
Requires further treatment to get metallic gold

Direct Electrowinning

Fast Reaction Kinetics
Fair Selectivity
Sensitive to TSS
Low Capacity
Can produce a product ready for melting

3.0 EVALUATION CRITERIA

In order to evaluate the various recovery processes on a more quantitative basis, evaluation criteria were established. These are based on criteria established by J. Dean Thibault and Associates and are contained in Table 3. (Also, see pages 16-22 of Appendix 2).

Mark	Criteria
1 and 2	not acceptable, or poor or high risk
3 to 5	average risk, identified limits but solution known
6 to 8	acceptable to good
9 and 10	high proven standards

Table 4 summarizes the marking for the five identified recovery processes. Evaluation of the various recovery processes favours the established procedures largely because the newer technologies are relatively untried and still have operating problems. The criteria are fairly general and so all the processes have values that are quite close to each other. The processes can be ranked in the order zinc precipitation, carbon adsorption, ion exchange, solvent extraction and direct electrowinning.

The evaluation sheets are included in Appendix 1 of Thibault and Associates Inc., report which forms Appendix 2 of this report.

TABLE 3

1.0 TECHNICAL FEASIBILITY (50%)

- 1.1 Feedstock Characteristics (25%)
  - 1.1.1 Solution Fouling (poisoning effect) (20%)
  - 1.1.2 Impurities and Selectivity of Operation (25%)
  - 1.1.3 Throughput Capability / Kinetics (10%)
  - 1.1.4 Metal Concentration Sensitivity (40%)
  - 1.1.5 Dependency on Up-Stream Processing (5%)
- 1.2 Operating Efficiency (45%)
  - 1.2.1 Metal Recovery and Grade (25%)
  - 1.2.2 Impurity rejection (20%)
  - 1.2.3 Complexity of Operating Variables (15%)
  - 1.2.4 Control of Steady State Mass Balance (10%)
  - 1.2.5 Control of Kinetics and Thermodynamics (105)
  - 1.2.6 Inventory or hold-up Capacity (10%)
  - 1.2.7 Downstream Processing Requirements (5%)
  - 1.2.8 Need for Recycle Streams (5%)
- 1.3 Raw materials and Consumable (15%)
  - 1.3.1 Energy Demand (35%)
  - 1.3.2 Reagent Consumption and Preparation (15%)
  - 1.3.3 Utility Requirements (steam, air, water) (25%)
  - 1.3.4 Availability, Source and Alternatives (15%)
  - 1.3.5 Replacement Parts / Materials (refractory, etc.) (10%)
- 1.4 Mechanical (15%)
  - 1.4.1 Specialization of Equipment (10%)
  - 1.4.2 Operator Friendly (25%)
  - 1.4.3 Fluid Transfer Capabilities and Storage (15%)
  - 1.4.4 Maintenance Requirements and Downtime (35%)
  - 1.4.5 Automation and Instrumentation Requirements (15%)

2.0 ECONOMIC AND INVESTMENT CRITERIA (30%)

- 2.1 Labour (15%)

TABLE 3 (CONT'D)

2.1.1 Operator Requirements (40%)
2.1.2 Maintenance Requirements (30%)
2.1.3 Contract Employees (20%)
2.1.4 Administrative Staff (10%)

2.2 Raw Materials and Energy (25%)

2.2.1 Cost to Purchase (40%)
2.2.2 Transportation Cost (30%)
2.2.3 Storage Cost (10%)
2.2.4 Utility Cost (20%)

2.3 Capital Investment (45%)

2.3.1 Cost to Purchase or Fabricate (30%)
2.3.2 Installation Costs (25%)
2.3.3 Transportation Costs (15%)
2.3.4 Plant Area Requirements (20%)
2.3.5 Duty and/or Tariffs (10%)

2.4 Sensitivity (15%)

2.4.1 Variation with Feed Throughput Tonnage (50%)
2.4.2 Variation with Feed Concentrations (50%)

3.0 DEVELOPMENT STATUS OF TECHNOLOGY (20%)

3.1 Support of Process Development (30%)

3.1.1 Documentation of Technical Development (50%)
3.1.2 Bench and Pilot Plant Studies (20%)
3.1.3 Commercial Optimization Studies (30%)

3.2 Availability of Technical Support (25%)

3.2.1 Patent(s) Rights (non-infringement) (50%)
3.2.2 Consultation by Developers (40%)
3.2.3 Consultation by Equipment Suppliers (10%)

TABLE 3 (CONT'D)

3.3 Commercial Applications (15%)

- 3.3.1 Number of Commercial Operations (40%)
- 3.3.2 Scale of Commercial Operations (20%)
- 3.3.3 Similarity in Feedstock Characteristics (20%)
- 3.3.4 Similarity in Process Application (20%)

3.4 Engineering Scale-Up Data (15%)

- 3.4.1 Design Variables (40%)
- 3.4.2 Physical and Chemical Data (30%)
- 3.4.3 Computer Simulation (10%)
- 3.4.4 risk Factors (20%)

3.5 Engineering Management (15%)

- 3.5.1 Experience in Construction Procurement (50%)
- 3.5.2 Extent of Engineering Required (25%)
- 3.5.3 Construction Contractors Requirements (25%)

TABLE 4

	weight	Zinc Precipitation	Carbon Adsorption	Ion Exchange	Solvent Extraction	Direct Electro-winning
Technical Feasibility	50%	34.20	26.08	27.59	28.60	22.46
Economics and Investment	30%	20.81	16.76	21.09	17.91	17.54
Development Status	20%	18.01	17.63	10.77	7.31	5.87
Overall Weighted Value		73.02	60.47	59.45	53.82	45.97

4.0 SUMMARY OF PREVIOUS TESTWORK

The Mineral Development Branch of the Department of Natural Resources and Energy since September 1987 have performed some leach tests on Mispec Resources ore.

Core samples were used for these tests. Rolls tests on samples crushed to 19 mm (3/4 inch) and -74μm showed that maximum recovery was achieved when using 2 kg/tonne and 1 kg/tonne NaCN respectively. Gold extraction from the -74μm samples was 98% after 48 hours.

Small scale column leach tests using core crushed to 19mm and then recrushed to 13 mm indicated that 42% of the gold was recovered at the coarser size and a further 43% was recovered at the finer size for a total recovery of 85%. Total leach time was 56 days; 35 days with the coarser material and 21 days after recrushing. Cyanide concentration was 2 g NaCN per litre of solution.

The last test campaign used a sample crushed to 80% passing 9.5 mm (1/3 inch). After 38 days leaching 57.6% of the gold was recovered. Cyanide consumption was 3 kg/tonne. Copper, zinc and iron concentrations in the leach liquor were as follows:

Days	Ions Concentration mg/L		
Leaching	Cu	Zn	Fe
10	180	120	150
25	375	175	225
38	640	320	350

Arsenic and antimony concentrations were 3 to 8 mg/L.

Particle size analysis of the crushed ore showed that high levels of copper occurred in the 5 to 7 mm fraction and in the fines. The above analysis indicates that this copper is soluble and that high levels of free cyanide would be required to complex this copper and have sufficient cyanide for gold leaching.

5.0 PRELIMINARY WORK

The ore to be used in the test programme came from Mispec Resources. An assay of a representative sample of the first bulk sample indicated that there was practically no gold in it. Therefore a second bulk sample was collected. This assayed by fire assay 0.6 g/t. A head assay of the sample gave the following (units are g/t except where noted otherwise):

Au	Ag	As	Cu	Sb	Zn	Fe,%
0.6	18.0	587.2	56.0	189.3	216.9	5.80

The bulk sample weighed approximately four tonnes. Approximately half the sample was dried and crushed to a nominal minus one inch. A sieve analysis of the coarse crushed material gave the following distribution:

Size, mm	wt%
+19	29.6
-19 +13	15.6
-13 +6	10.8
-6	44.0
	100.0

From the distribution it can be seen that the ore contains a large percentage of fine material. Visually, much of the rock is very friable.

Analysis of the size fractions indicates that most of the gold is in the -6 mm fraction.

Size, mm	Au, g/t	Au Distribution, %
+19	0.225	9.9
-19 +13	0.161	3.7
-13 +6	0.289	4.6
-6	1.254	81.8
	0.675	100.0

6.0 MINERALOGICAL EXAMINATION

The findings of the mineralogical examination have been reported in Interim Report No. 1. The full mineralogical report on the examination of Mispec ore is included in Appendix 1. Core samples from each zone as well as the bulk samples were analyzed.

The examination of the bulk sample indicated that it is more oxidised than the core material. The host rock is quartz with lenses and veinlets of illite. Pyrite has been highly altered to geothite. Rutile was observed. In addition, there is a fine-grained mixture of friable illite, quartz, iron oxides and hydroxides, goethite and possible carbonates. No gold was detected.

Gold was observed in two zones - the Birches Zone and the Road Zone. It was predominantly free native gold with some gold tellurides observed in the Road Zone sample.

Copper and iron sulphides were observed and these may solubilize in the gold leachant thus interfering with the recovery of the gold. Some Cu and Cu-Fe sulphates are present in the Birches Zone. If they are soluble, the copper and iron may interfere with the gold recovery by cyanidation and the sulphate ion is detrimental when using thiourea as a leachant. Carbonates were observed in all zones except the bulk sample. While they are beneficial for leaching with cyanide, they are detrimental when using any acid leach process. An acid pre-wash would probably solve the problem but may not be very practical for a heap leach operation.

Illite is likely to present percolation problems in that it is likely to block solution channels in the heap and may eventually form an impermeable barrier at the bottom of the heap.

7.0 BENCHSCALE TESTWORK

The primary objective of the benchscale testing was to provide data for the heap leach simulation in the column. The tests provided data on the effects of crush size, cyanide solution strength and dosage and reaction kinetics. Also, comparisons of leachants were made and some other miscellaneous tests were run. These tests were designed to augment the work already done by DNRE.

The leaching tests were run using either a standard bottle roll or a small column. In the bottle roll test, a known weight of solids are pulped with water at approximately 33% solids in a bottle. The pH is adjusted to the required point with lime and cyanide is added to give an initial 0.05% NaCN solution. The bottle is rolled for the required length of time, usually 24 or 48 hours. NaCN content and pH are checked after one hour, six hours and 24 hours and more lime and NaCN are added as necessary to maintain the starting values. At the end of the leaching time, the pulp is filtered and the filter cake is washed, dried and weighed. The filtrate and filter cake are assayed for gold and other elements as required.

For the small column tests, a two foot length of 2 inch plastic pipe was used. Mesh was attached to the bottom end to hold the sample in place. A known weight of sample was placed in the column and the cyanide solution was cycled through the column. At the end of the test the filtrate was analyzed.

7.1 Bottle Roll Tests

The first set of tests involved leaching duplicate samples of the ore after grinding to 100% -73μm and leaching various size fractions, both as is and ground. These results are summarized in Table 5. These results indicate that the material leaches quite readily and that high recoveries can be obtained. Recoveries over 48 hours for the ground samples exceeded 80%. Cyanide consumption ranged from 1.5 kg/t to 1.7 kg/t. The filtrates from Tests 3 to 6 were analyzed for various elements and these results are given in Table 6.

TABLE 5
RESULTS OF INITIAL LEACH TESTS

Test	Sample	Au g/t Feed	Residue	Au Recovery,%	NaCN Consumption g/t	Leach Time Days
1	Bulk Sample (-73μm)	0.62	0.03	96.2	500	2
2	Bulk Sample (-73μm)	0.62	0.09	84.5	550	2
3	-13 mm + 6 mm ("as is")	0.78	0.28	64.1	1680	7
4	-13 mm + 6 mm (-73μm)	0.78	0.03	96.2	1660	2
5	-19 mm + 13 mm ("as is")	0.16	0.03	81.2	1533	7
6	-19 mm + 13 mm (-73μm)	0.16	0.03	81.2	1500	2

TABLE 6

RESULTS OF FILTRATE ANALYSES, mg/L

Test No.	3	4	5	6
Ag	25.1	202.9	7.3	284.9
Au	0.95	0.72	0.60	0.05
Cu	2.7	3.32	1.88	2.12
Fe	0.95	3.70	41.67	1.30
Hg	<.001	<.001	<.001	<.001
Pb	<.1	<.1	<.1	<.1
Sb	0.36	0.94	0.30	0.61
Zn	0.55	0.36	0.65	0.88

These assays show that interfering ion levels are low e.g. the maximum copper value was 3.32 mg/L. An interesting result is for silver which shows a much higher value in solution from the ground ore. This indicates that without fine grinding the silver is not readily leached.

A comparison was made between NaCN and $Ca(CN)_2$. The samples were ground to 100% passing 73μm. The cyanides were added so that the CN^- in solution was equal for both tests.[1] Analysis of the filtrates gave the following results:

Assays mg/l

	Au	Ag	As	Cu	Fe	Hg	Pb	Sb	Zn
NaCN	0.69	284.9	2.0	1.9	0.2	<.001	<.1	1.5	.08
$Ca(CN)_2$	0.68	255.9	1.1	2.0	15.6	<.001	<.1	2.7	0.6

Since the gold in solution is greater than the gold theoretically available in the sample, accurate gold recoveries cannot be calculated. In earlier tests using the same material and having similar solutio tenors, the gold recovery was approxiamtely 96%.

It has been theorized that $Ca(CN)_2$ dissolves less arsenic and antimony than NaCn. The levels of both ions in solution, however, are low and do not present a problem. The results do not show any significant advantages of one type of cyanide over the other. There may be economic benefits in using one cyanide versus the other depending upon the prices delivered to the minesite. NaCN was used for the rest of the cyanidation testwork.

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(1) Since commercial sodium cyanide assays approximately 98% NaCN while commercial calcium cyanide assays approximately 49% NaCN, it is necessary to add twice as much calcium cyanide as sodium cyanide.

7.2 Thiourea Testing

The mineralogical examination had shown that the sample was relatively high in carbonates. This means that acid consumption would be very high for any process requiring low pH conditions. Approximately 100 g each of -19 mm + 13 mm and -13 mm + 6 mm material were placed in beakers, and acid to maintain pH3 was added over seven days. The following results were obtained:

	Sample Weight Loss, %	H_2SO_4 Consumption, kg/t
-19 mm + 13 mm	18.5	38
-13 mm + 6 mm	19.4	39

From these findings it can be seen that acid consumption is high and that probably large quantities of $CaSO_4$ would be formed which would tend to block flow channels in the heap.

Some bottle roll tests were run using thiourea and gound ore. The pH was maintained at 1.5 with H_2SO_4 and Fe_3SO_4 was added as a catalyst for the thiourea leaching. One sample was run for nine hours and the other ran for 24 hours.

The results are presented in Table 7. Compared with earlier results with cyanide, lead, zinc and antimony values in the leach solution are higher while silver and arsenic values are lower. Iron is considerably higher because of the ferric sulphate addition. With the exception of iron, all the values for the various ions in solution are low. Unfortunately gold recovery cannot be calculated because the amount of gold theoretically available in the sample is less than 1 mg and the assay limit of detection is only 1 mg/L.

Based on the above findings thiourea does not appear to be a suitable leachant for the Cape Spencer ores.

TABLE 7

RESULTS FROM THIOUREA LEACHING, mg/L

Elements	Sample T1 (after 9 hours)	Sample T2 (after 9 hours)	Sample T2 (after 24 hours)
Au	<1.0	<1.0	<1.0
Ag	<.5	<.5	<.5
As	<.05	<.05	<.05
Cu	9.0	8.4	2.4
Fe	4571	4352	1700
Pb	9.0	8.5	2.2
Sb	6.8	6.4	1.8
Zn	26.3	25.2	10.9

7.3 Laboratory Percolation Tests

Some preliminary percolation tests were run to investigate particle size versus percolation. Solutions were not assayed. These tests showed that percolation was acceptable at the nominal 25 mm size, that is fines did not block the flow and the flow rate was acceptable over time. With finer particle sizes the amount of fines was relatively higher and blocking occurred. Based on these findings, it was decided to use the nominal 25 mm material in the large scale column.

Tests were conducted to investigate the effects of different strength cyanide solutions. Solutions containing 0.5, 1, 2 and 5 g/l NaCN were cycled through the laboratory percolation column. Ore sample particle size was nominally 25 mm. The results from these tests are contained in Table 8.

The results show that as the cyanide solution strength increases the rate of gold dissolution increases and the total amount of dissolved gold increases. Using 0.5 g/L NaCN solution produced a dissolved gold content of 60 mg/L while the 5 g/L NaCN solution contained 519 mg/L. Effects on other metal ions are varied. Silver is unaffected probably because its dissolution is so low while copper, zinc and iron show decreases in their content in the leach liquor possibly because of complexing with the cyanide or because of subtle variations in the ore sample. The decrease in dissolved copper is particularly high, going from 74.5 mg/L at 0.5 g/L NaCN to 3.1 mg/L at 5 g/L NaCN.

TABLE 8

RESULTS OF PERCOLATION TESTS AT VARIOUS NaCN STRENGTHS

Solution Strength g/l	ASSAYS mg/l Au 24hr	Au end	Ag 24hr	Ag end	Cu 2hr	Cu end	Fe 24hr	Fe end	Zn 24hr	Zn end
0.5	----	60	----	<0.5	----	74.5	----	29.0	----	6.5
1	<30	<30	<0.5	<0.5	84.4	61.2	24.9	8.4	16.5	11.4
2	----	312	----	<0.5	----	6.2	----	13.9	----	1.6
5	162	519	<0.5	<0.5	1.7	3.1	9.6	10.2	0.5	1.2

8.0 PILOT SCALE COLUMN TESTING

The pilot scale column is approximately 30 cm in diameter by approximately 4.5 m high. At the bottom it is flanged and it has a drainage pipe with a valve. The pipe sticks up above the flange inside the column and has 6 mm holes every 6 mm for drainage. The feed is introduced to the column from the top. Leach solution is pumped from the sump to the top of the column, percolates down through the column and drains back into the sump.

Approximately 300 kg of sample were placed in the column. The particle top size was nominally 25 mm. Approximately 175 litres of solution containing 2 g/l of NaCN was used as the leach solution. Solution pH was monitored continuously and was connected to a low level alarm. Cyanide strength was monitored intermittently. Lime and NaCN were added as required.

The column was operated for 40 days. Samples of the solution were collected at intervals and assayed. The results are presented in Table 9 and Figure 4. From the table it can be seen that copper and iron increase over time but that the values are still very low by the end of the leach. The highest values were 4.2 mg/L for copper and 3.9 mg/L for iron. From the figures it can be seen that the initial dissolution rate over the first two days is quite high and is almost constant over the rest of the time. Based on a head assay of 0.62 g/t the gold content of the ore in the column is 0.186 g and assuming 150 litres of solution (allowing for some losses), the gold content of the pregnant liquor theoretically is 1.24 mg/litre. Therefore, with an assayed end value of 1.5 mg/L of gold in solution, it is probable that the gold recovery was approximately 100% after 40 days. After 24 days 81% of the recoverable gold (based on the assays) had been leached.

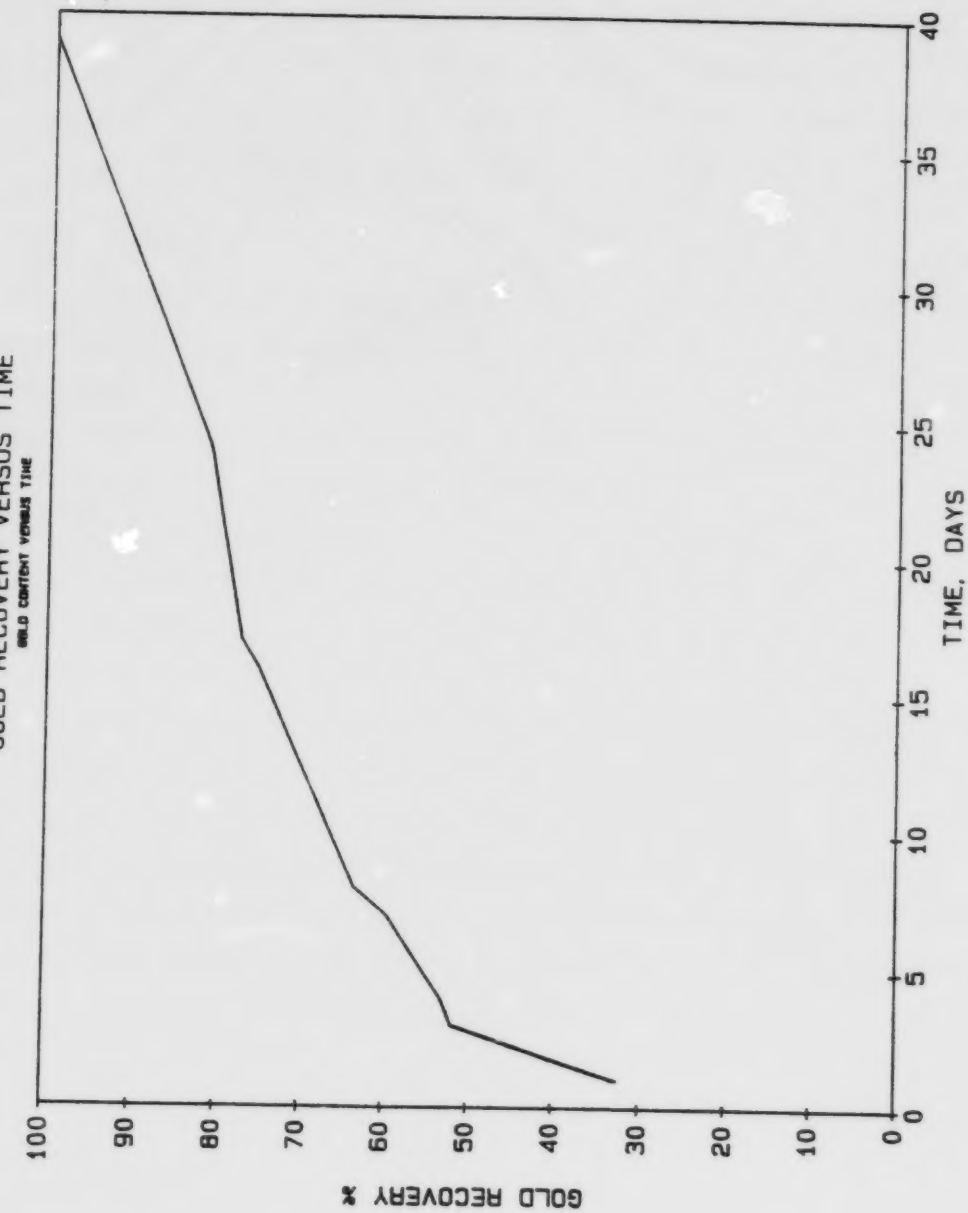
TABLE 9

RESULTS OF COLUMN LEACH

	Assays mg/L			Recovery, %
Days	**Au**	**Cu**	**Fe**	**Au**
1	0.50	2.0	3.3	32.5
3	0.80	2.6	1.2	51.9
4	0.82	2.6	1.7	53.2
7	0.92	2.8	1.5	59.7
8	0.98	2.9	1.8	63.6
16	1.16	3.2	2.4	75.3
17	1.19	3.3	2.4	77.3
18	1.18	3.3	2.5	76.6
24	1.25	3.5	3.9	81.2
39	1.54	4.2	3.4	100.0

FIGURE 4

GOLD RECOVERY VERSUS TIME



9.0 RECOVERY PROCESSES

In the course of the literature survey, five potential recovery processes were identified: zinc precipitation (Merrill-Crowe), carbon absorption, ion exchange, solvent extraction and direct electrowinning. The evaluation of these processes indicated that zinc precipitation and carbon absorption were the two best techniques. In addition, since the testing was being carried out at bench-scale, it was felt that it would be informative to include solvent extraction and direct electrowinning in the programme. Ion exchange was not included because of problems finding a suitable resin.

All the tests were conducted using the pregnant liquor produced in the column heap leach simulation. All the tests except the direct electrowinning tests were conducted in the laboratories at RPC. The direct electrowinning tests were conducted in a small test cell at the Mineral Development Branch laboratory.

9.1 Zinc Precipitation (Merrill-Crowe)

For these tests approximately 2 litres of solution were filtered into a Buchner flask. The flask was made airtight and connected to the vacuum system to deoxygenate the sample. A magnetic stirrer was used to agitate the sample. After approximately two hours zinc dust and lead nitrate were added. After a short mixing time an aliquot was filtered and the filtrate was assayed for gold, copper and iron. For some of the tests copper sulphate was added to try to increase the Cu ion content of the sample.

The results of these tests are given below:

Test No.	Zinc Dust, g	Cu Added	Assays mg/L Au	Cu	Fe	Au Recovery %
Z1	0.1	No	0.54	5.5	5.3	66
Z2	1.0	No	0.25	7.9	5.3	84
Z3	0.1	Yes	0.76	64.4	4.0	52
Z4	0.5	Yes	0.43	65.2	5.3	73
Z5	0.7	Yes	0.52	61.3	5.3	67
Feed	---	No	1.585	4.7	4.0	---

These results appear to indicate that the efficiency of zinc precipitation is low at low gold concentrations. Statistical analyses indicate that there is a high correlation between the amount of zinc dust added and gold recovery. The calculated gold content of the sample is 3.17 mg; therefore, the amount of zinc dust and lead nitrate required to achieve good recoveries is well in excess of the accepted standard of 2:1 Zn:Au. Higher copper levels do not appear to have affected the gold recovery.

9.2 Carbon Absorption

A small carbon column was erected in the laboratory to carry out three carbon absorption tests. The column was made from a 1 litre graduated cylinder with the bottom removed and nylon mesh held the carbon in place. Approximately 250 g of activated carbon were placed in the column and approximately 18 litres of pregnant solution were cycled through the column for a total of 20 times.

The results of gold remaining in solution are presented in the following Table:

Number of Cycles	Residual Au, PPB	Au Recovery, %
1	1585	---
3	270	83
5	75	95
8	50	97
11	55	97
15	55	97
20	50	97

These results show that carbon absorption gave excellent recovery even with a low gold content in the feed. The carbon gold loading was 63.2 g/t which is approximately one tenth of the typical loadings which can be achieved with carbon.

9.3 Ion Exchange

Experiments were not conducted with ion exchange resins. Although in theory they can be used there are still many practical problems with them. Advantages noted for ion exchange resins are that they are not as readily poisoned by organic matter or fines and that the loading rates are high compared with carbon. Disadvantages are poor selectivity, slow elution rates and relatively high costs. Recent work, particularly at Mintek in South Africa (34) has moved away from weak base resins and concentrated on strong base resins. Testing has shown that problems related to screening and stability are unfounded and recent work particularly at the Golden Jubilee Mine has shown improvements in elution. Even in the advanced work on developing much more selective resins at Mintek the results are still very inconsistent (34).

Based on the above arguments is was decided that since it was not possible to obtain a sample of Mintek's resin there was no advantage in pursuing this area further at this time.

9.4 Solvent Extraction

Modifications of the standard analytical procedure for gold determination were used to investigate solvent extraction. The extractant used was diisobutyl ketone (DIBK). Four 100 ml samples of the pregnant liquor were taken and 0, 10, 20 or 30μg of gold were added to the respective aliquot. The gold was extracted with DIBK and then analysed by atomic absorption spectroscopy.

The results were as follows:

	Gold Extracted, μg	Gold Added, μg	Analyzed Additional Gold, μg
S1	71.7	0	0
S2	81.4	10	9.7
S3	92.9	20	21.2
S4	101.8	30	30.1

These results show that extraction generally is complete. This testing was limited, therefore, more work particularly at a bench-scale continuous level needs to be carried out to study different extractants and other parameters.

9.5 Electrowinning

To investigate direct electrowinning tests were run in a small electrowinning cell which held approximately 1 litre of solution. The solution was filtered prior to testing. For the first test, stainless steel cathodes and anodes were used. DC power input was 7 volts at 5 amps. In the second test a cathode was constructed by making a bag from nylon mesh and filling it with tightly packed granular carbon. The starting electrical inputs were 10 volts at 2 amps and the final values were 11 volts at 1 amp.

The results of the tests are given below:

	Assays, mg/ml		
	Au	**Cu**	**Fe**
Feed solution	0.76	4.7	6.7
Test No. EW1 final solution	0.22	3.9	10.7
Test No. EW2 final solution	0.36	7.1	8.0

These results indicate that Test EW1 and EW2 gave 71% and 53% Au recoveries respectively. These are not as good as the recoveries for other processes but do indicate that more work should be carried out in this area. A concern in Test EW2 is how much of the gold recovery is due to carbon adsorption and how much is due to electrowinning.

10.0 DISCUSSION OF RESULTS

The results show that the material represented by the sample used in this testwork is easily heap leached and that very high gold recoveries (greater than 95%) can be achieved. Compared with work done previously on drill core material, higher gold recoveries at a coarser granulometry were obtained. Further, the sample was very low in potentially interfering ions i.e. copper, zinc, arsenic, antimony and iron whereas in the earlier work the pregnant solution contained 640 mg/L of copper after 38 days leaching.

Bench scale testwork demonstrated that NaCN and $Ca(CN)_2$ gave similar gold extractions (~96%) while reagent consumptions with thiourea were very high. Using thiourea on this ore did not appear to offer any advantages. Other leachants were not tested. As with thiourea, low pH systems would not be practical or economic because of the high acid consuming components of the ore. Further, in a heap leach milieux, gaseous leachants would not be practical. This leaves cyanide and thiosulphate as the only potential candidates and because the ore responded so easily to cyanidation, cyanide would probably be the most economic leachant. Additional bench scale testing demonstrated that leaching could be done with a nominal top size of 25 mm and that dissolution rates were much higher for 5 g/L cyanide solutions than for 1 g/L cyanide solutions.

Column testing confirmed that the gold was readily leached. Gold recovery after 40 days was 100%. Flow rate through the column was approximately 2.9×10^{-3} $m^3/min/m^2$ which indicates that plugging was not a problem. Also, the high gold recovery indicates that channelling did not occur.

Bench scale tests were carried out for zinc precipitation, carbon absorption, solvent extraction and direct electrowinning. Carbon absorption was able to remove 97% of the gold from the pregnant solution while solvent extraction appeared to recover approximately 100% of the gold in solution. Recovery of gold using zinc precipitation was, at best, 84% and this was after adding 500 mg/L of zinc dust which is 79 times the accepted industry standard. Other work has demonstrated that the efficiency of zinc precipition declines with

decreasing solution tenors. It may be possible to lower the zinc usage in plant practice by passing some of the deaerated solution directly through the final filter without adding zinc dust.

Direct electrowinning resulted in 71% gold recovery. These results are very preliminary but are very encouraging. No optimization was tried. Also, the possible breakdown of cyanide in the solution was not investigated.

11.0 PRELIMINARY PROCESS ECONOMICS

As part of the project, capital and operating costs for a 250,000 tonne per annum heap leach operation have been generated. The estimated fixed capital investment for an operation using either zinc precipitation (Merrill-Crowe) or carbon adsorption is approximately $1.42 million. These costs are ± 30% and are first quarter 1990 dollars. Table 10 contains a more detailed description of the direct and indirect costs. The costs include engineering, construction, and supply and installation of necessary equipment for the recovery of gold and the production of dore. Site preparation, mining, crushing, grinding, heap preparation and heap removal costs are not included.

The estimated annual operating costs are $0.76 million for the zinc precipitation option and $0.78 million for the carbon adsorption option. A more detailed summary of these costs is given in Table 11. The major difference between the two processes is that it was assumed that 12 operators would be required for the carbon adsorption process while nine operators would be needed for the zinc precipitation process.

Equipment sizing was based on information generated by a water balance computer programme developed by Thibault and Associates. This balance used the following criteria:

solution application rate	169 l/h/m_2
volumetric flow to recovery plant	65 m^3/h
gold content of pregnant solution	0.36 mg/m^3
recycle solution rate	342 m^3/h
gold head grade	0.6 g/t
corrected heap area (assuming 2 lifts of 3.67 m each)	30,910 m^2
leach operation	340 days/year

Where applicable, test data generated in the course of this project were used as preliminary design criteria. Otherwise, published information and/or generally accepted engineering practices were used. Design criteria developed by Thibault and Associates Inc. for zinc precipitation and carbon adsorption are given in Tables 12 and 13, respectively. Figures 5 and 6 show the relevent preliminary process diagrams. Back-up data and the figures are included in Appendix 2 which contains Thibault and Associates Inc.s' report.

TABLE 10

ORDER OF MAGNITUDE CAPITAL COST

DIRECT COSTS (DC)	**Zinc Precipitation**	**Carbon Adsorption**
Equipment (Installed)*	$ 657,400	$ 556,000
Process Building and Site Preparation	175,000	247,500
Piping and Installation	21,500	27,000
Instrumentation Supply and Installation	45,000	97,500
Electrical Supply and Installation	45,000	30,000
TOTAL DIRECT COST	**$ 943,900**	**$ 958,000**
INDIRECT COST		
Material Indirects (8% DC Materials)	$ 30,500	$ 23,900
Engineering, Design Only	200,000	200,000
Construction Management (12% DC Labour)	33,000	30,800
Equipment and Field Changes (2% DC)	19,000	19,200
Project Contingency (20% DC)	189,000	191,600
TOTAL INDIRECTS	**$ 471,500**	**$ 465,500**
FIXED CAPITAL INVESTMENT	$1,415,400	$1,423,500
WORKING CAPITAL (10%)	$ 141,500	$ 142,400
TOTAL CAPITAL INVESTMENT	**$1,556,900**	**$1,565,900**

* Based on trades mix and includes transportation FOB Saint John.
(Probability of estimate being +/- 10% is 40% to 50%).

TABLE 11

ORDER OF MAGNITUDE ANNUAL PRODUCTION COST

			Zinc Precipitation	Carbon Adsorption
I	**DIRECT COST**			
	A.	Raw Materials	$100,000	$ 41,000
	B.	Labour	132,500	194,500
	C.	Direct Supervision	35,000	35,000
	D.	Utilities	22,000	38,500
	E.	Maintenance (2% FCI)	28,300	28,500
	F.	Supplies (0.5% FCI)	7,000	7,000
	G.	Laboratory Charges	65,000	65,000
II	**FIXED CHARGES**			
	A.	Depreciation (10% FCI)	141,500	142,300
	B.	Taxes (8% Revenue)	140,800	140,800
	C.	Insurance (0.6% FCI)	8,500	8,500
	D.	Overhead (90% Labour)	39,750	39,750
III	**GENERAL EXPENSES**			
	A.	Administration (10% Labour)	13,000	13,000
	PRODUCTION COST		**$733,350**	**$753,850**
CONTINGENCY (3%)			22,000	22,600
	TOTAL PRODUCTION COST		**$755,350**	**$776,450**
	COST PER TONNE ORE		**$ 3.02**	**$ 3.11**

TABLE 12
DESIGN CRITERIA FOR ZINC PRECIPITATION

CLARIFICATION

Solid Settling Area in Pregnant Pond	0.5 to 1.5 m^2/t/day
Solution Filtration Rate	3.7 $m^3/hr/m^2$
Bulk Density of Clays Filtered	1.19 t/m^3
Bulk Density of Diatomaceous Earth (DE)	0.317 t/m^3
Precoat Thickness of DE	1.5 yo 3.0 mm

DEAREATION

Vacuum	558 to 635 mm Hg
Dissolved Oxygen Concentration	8 mg/L less than 1 mg/L
Vacuum Capacity	142 to 217 cm^3/ sec/ lt solution trated

PRECIPITATION

Zinc Addition	1-2 g/g Au
Lead Addition	40 g lead nitrate per 450 g Zn
Diatomaceous Earth	25% to 50% of Zn consumed by weight
Precipitate	10% to 40% Au

TABLE 13
DESIGN CRITERIA FOR CARBON ADSORPTION

CARBON COLUMN

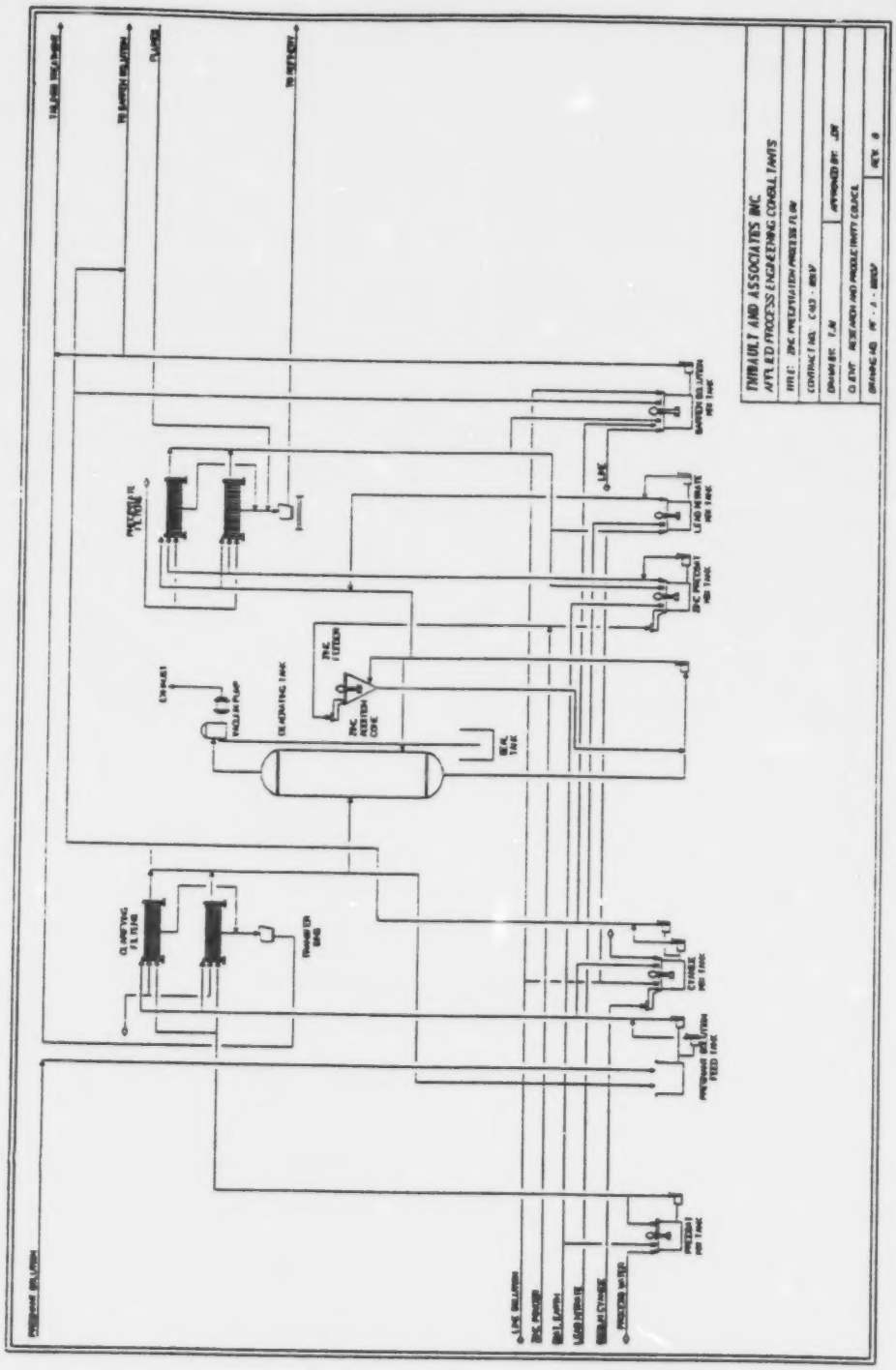
Superficial Solution Velocity	60 m/hr
Height to Diameter Ratio	6.5:1
Carbon Maximum Loading of Gold	6,500 g/t
Number of Columns	1

STRIP VESSEL

Strip Solution	0.1% NaOH, 0.1% CN
Temperature	130°C
Flow Rate	2 to 4 bed volumes per hour
Pressure	Ambient to 25 psi
Strip Time	70 hours
Height to Diameter Ratio	6:1 to 10:1
Superficial Solution Velocity	7 m/hr

CARBON REGENERATION

Activation Temperature	650°C
Residence Time	10-15 minutes
Kiln Loading Basis	11 kg/hr/m^2
Feed	50% Carbon Slurry


THIBAULT AND ASSOCIATES INC
APPLIED PROCESS ENGINEERING CONSULTANTS
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CONTRACT NO. C-103 - REV
DRAWN BY: TJM
APPROVED BY: JDT
CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL
DRAWING NO. PF - 1 - REV
REV. 0

FIGURE 5

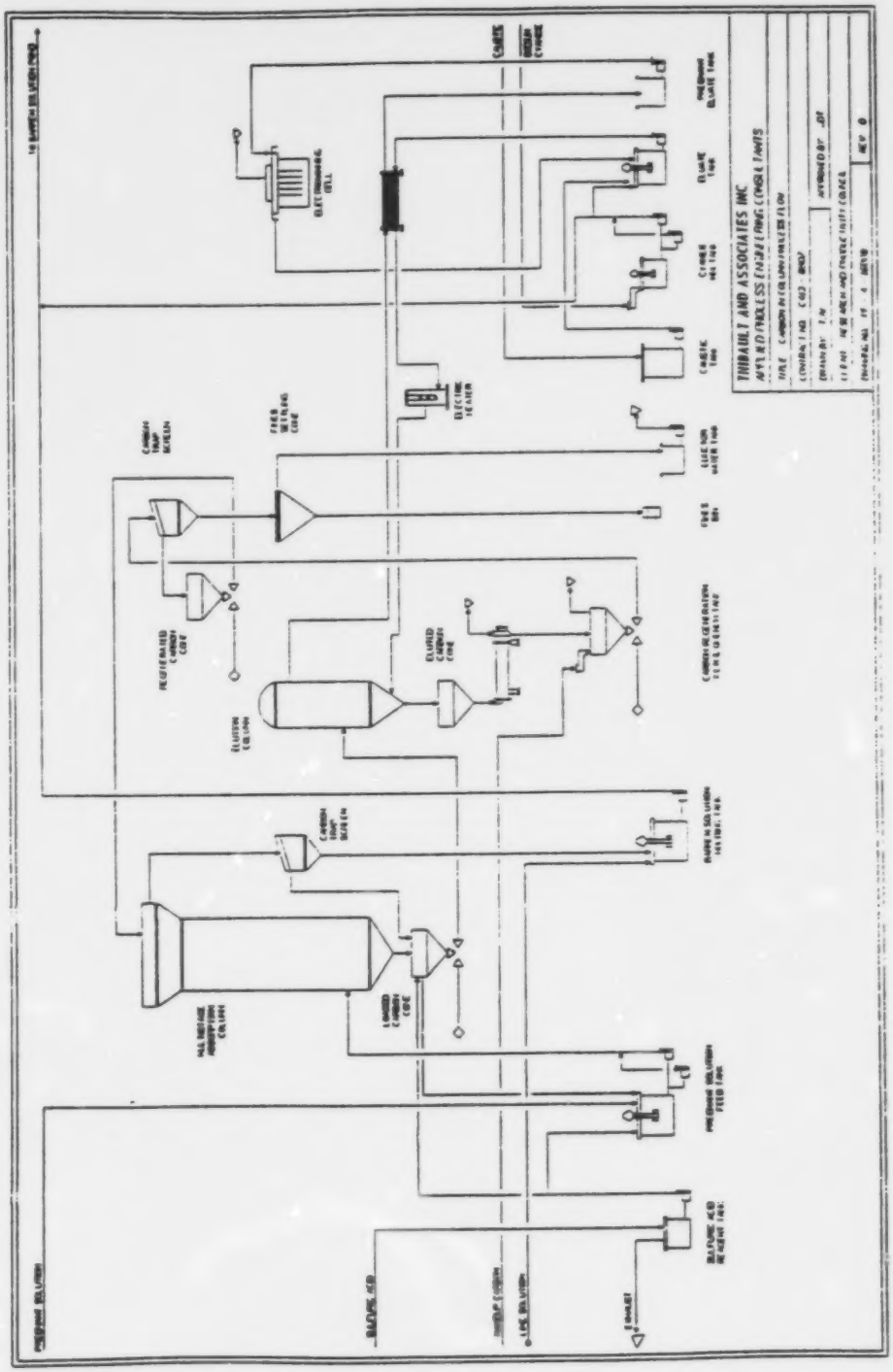

FIGURE 6

12.0 CONCLUSIONS

The literature survey identified five potential technologies for recovering gold from heap leach pregnant liquors. These processes are zinc precipitation (Merrill-Crowe), carbon absorption, ion exchange, solvent extraction and direct electrowinning. A theoretical evaluation of these processes for their desirability in a plant ranked them in the order listed above. All except ion exchange were evaluated in the laboratory using a pregnant solution generated in a simulated heap leach.

The ore sample used in the test programme was provided by Mispec Resources from their Cape Spencer deposit. The ore assayed 0.6 g/t. The mineralogy of the deposit indicates that it is predominantly quartz with lenses and veinlets of illite. Gold is predominantly free but some gold tellurides were observed. The material used in the leaching tests was more oxidized than core material. Copper and iron sulphides were observed.

Preliminary testing showed that the ore sample was easily leached and that NaCN was the best leachant. Thiourea and other acid milieu leachants were contraindicated because of the high acid consumption due to the carbonate content of the ore. Leaching the ore at a nominal 25 mm size gave good recoveries.

Laboratory testing of the various recovery processes showed that carbon absorption gave 97% recovery of the gold from solution; zinc precipitation gave 84% gold recovery; solvent extraction gave 100% gold recovery; and the direct electrowinning gave 71% gold recovery. Zinc precipitation is relatively inefficient with low solution tenors.

The estimated cost for a 250,000 tonne per annum heap leach operation is $1.42 million and the estimated annual operating costs are $0.76 million for zinc precipitation or $0.78 million for carbon absorption.

The results of this study show that, based on the sample used, a viable heap leach operation could be carried out provided sufficient reserves are available. The study indicates that an operation with a heap approximately three metres high and having a particle top size of approximately 25 mm and leaching with 2 g/L NaCN solution and using carbon absorption to recover the gold from solution will recover approximately 97% of the gold prior to electrowinning.

13.0 RECOMMENDATIONS

The results obtained during this project indicate that a heap leach operation is technically and probably economically viable. The next logical step following laboratory testwork would be a pilot scale leach. A heap containing approximately 1000 tonnes would be roughly one days requirements at 250,000 tonnes/year. One thousand tonnes stacked to three metres high would require a pad area approximately 18 m x 18 m.

Based on the results, carbon absorption is the best method for recovering the gold from solution. This was the technique eventually used by Gordex in their Cape Spencer operation. Also, this technique would be better able to handle any variations in solution gold content which might vary due to changes in leach parameters.

Prior to a pilot scale operation, however, it is recommended that more laboratory testing be carried out. A comparison of this work with that done previously indicates that there are some areas which may not leach as easily and which may be harder to process due to higher levels of interfering ions. Further laboratory (bottle roll) tests on a wider range of samples should better determine the overall leaching responses of the deposit.

During the laboratory tests, it was observed that silver recovery was very much higher at the finer grinds compared with the size distribution used in the simulated heap leach. It is recommended that some laboratory tests should be run using a finer grind and agglomeration to determine whether the silver recovery can be enhanced. In addition, this could give faster gold leaching rates which could improve the cash flow and the overall economics. Finally, this technique is able to ameliorate problems caused by clays.

If a finer grind is more advantageous, it might be useful to check out in the laboratory some other processing techniques such as traditional vat leaching with CIP or coal gold agglomeration. Other areas which should be checked, include using a flowsheet similar to Nova Gold's Murray Brook operation which treats 1000 tonnes per day in an indoor vat leach and is thus able to operate year round. Also, further developments in ion

exchange, solvent extraction and direct electrowinning should be monitored since all these techniques show considerable potential and may eventually offer advantages over carbon absorption and/or zinc precipitation.

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APPENDIX 1

MINERALOGICAL REPORT

MINERALOGY OF MISPEC ORE SAMPLES

Prepared for:

Department 8-0

Attention: Richard May

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Minerals & Fuel Technology

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1.0 SUMMARY

As required by RPC proposal MDPP/88/30, samples from high grade areas were collected for a mineralogical study. All sections were examined optically. Five of them were also examined using A JEOL 733 electron microprobe to identify unknown minerals and confirm identifications of other minerals. In addition four samples were examined specifically for gold-bearing phases using the electron microprobe interfaced with a KONTRON automated image analyzer.

In the samples, mineral assemblages vary little despite marked variation in the rock textures. Quartz, illite, ankerite, rutile and pyrite +/- albite or potassium feldspar are present in varying amounts in all the samples. Muscovite, chalcopyrite, galena, hematite, magnetite, sphalerite, Cu/Fe sulphates, bornite, covellite, digenite, and carrollite ($Cu\ Co_2\ S_4$) are present in some of the samples.

Gold-bearing minerals were observed in only two of the four sample sets examined - the Birches Zone and the Road Zone. In the Birches Zone samples, the gold is present as native gold particles (with <8% Ag) which range in size from <1 to 45 microns in size and are 80% by volume larger than 20 microns. These particles primarily occur as roughly equant grains associated with pyrite either as inclusions in or particles adjacent to the large (1-5 mm) poikiloblastic grains.

In the Road Zone gold occurrences were observed in one sample only. In this sample the gold is present as native gold (<7% Ag) and a gold-bearing telluride (26% Au, 41% Ag, 33% Te). The particles range in size from <1 to 58 microns and are 80% by volume larger than 11 microns. They generally occur as equant to elongate particles in and around a sphalerite/galena vein 2-5 mm thick. Many of them occur at interfaces between various minerals. The gold-bearing telluride grains are locally intergrown with the native gold.

To expose the gold-bearing minerals it will be necessary to grind the ore to approximately 60 mesh (250 microns). However, for the Birches Zone ore this value may be too high since much of the gold occurs in coarse-grained pyrite.

Numerous minerals present in these samples may affect the leaching and recovery of the gold. The abundant carbonate and much less common copper/iron sulphates will create problems if thiourea is used as a leachant. The various copper and iron sulphides may interfere with gold recovery from the pregnant solutions. The tellurides in the Road Zone sample set will make variations in the treatment necessary. And, finally, the presence of abundant illite will cause percolation problems.

Although many gold-bearing grains were observed, it is recommended that more samples be examined to increase the population studied and, therefore, improve the statistical results. Because a large percentage (by volume) of the gold is present in coarse grains gravity concentration would be an effective way to significantly increase the number of gold-bearing grains in each sample.

In addition, testing a bulk sample that better represents the ore is recommended. Collecting a sample from drilled material or from deeper trenches would ensure a better sample. The information in this report would be more relevant to this type of material since most of the samples examined were not oxidized to the extent that the bulk sample material was.

Finally, testing grinds, taking into consideration those proposed in the report, is recommended. The presence of fine-grained illite in all the samples should also be considered.

2.0 INTRODUCTION

This report is in fulfilment of the task set out in the mineralogy section of RPC proposal MDPP/88/30, "A Study of the Processing Options for Gold Recovery of Cape Spencer Ores". It contains a description of the host rocks and the gold occurrences observed. These descriptions of gold occurrences include grain size, grain shape, association of the grain, general liberation size of host particles (where applicable), and chemical composition of the gold phases. In the host rock descriptions grain size is referred to as "coarse, medium, and fine". The size ranges these refer to are >500 microns (0.5 mm), 20-500 microns, and <20 microns, respectively.

The samples were collected from three different zones which are referred to in the report - the Birches Zone, the Pond Zone, and the Road Zone. Their locations are given in Appendix A-1. According to Watters (personal communication) ore-grade gold in the Birches Zone occurs mainly in pyrite rich, intensely illitized and mylonitized granitoid rocks of the Millican Lake Intrusions where they contain an abundance of early specularite veins. Gold also occurs in (1) the intensely illitized mafic volcanic rocks within the same mylonitized zones in the Millican Lake Intrusions (2) within albitized microgranite (3) in adjacent, fault-emplaced illitized sedimentary rocks of the Cape Spencer Fm. The albitic alteration overprints illitic alteration in the Birches Zone and the gold may have been introduced during illitic alteration.

The Road Zone ores occur in an S_2-parallel zone of pyrite-rich, illitized and faulted sedimentary rocks of the Cape Spencer Formation lying overtop the northwest-dipping flank of the Precambrian granitoid. The deposit is exposed by erosion at its north end and plunges to the south-southwest, becoming a blind deposit beneath a D_2 thrust sheet of granitoid rock. The bulk sample was collected from this zone.

Gold in the Pond Zone also occurs in pyrite-rich, illitized rocks (Cape Spencer Fm sediments and granitoid mafic volcs of the Millican Lake Intrusions) where granitoid rocks are thrust over Cape Spencer Fm sedimentary rocks. Gold is more or less restricted to the sedimentary slice and the overlying granitoid slice.

3.0 METHOD OF ANALYSIS

Twenty-three polished thin sections were prepared from the core samples collected in May and the bulk sample collected in June. A list and description of the samples is given in Table 1. Each thin section was examined optically and briefly described. Five of these were examined using a JEOL 733 electron microprobe to identify unknown minerals or confirm optical identifications of various minerals.

To characterize the gold occurrences, four thin sections were examined using a KONTRON automated image analyzer interfaced with the electron microprobe. The gold was identified and various properties were measured or noted as follows:

(i) surface area of grain (sq. microns)
(ii) perimeter of grain (microns)
(iii) maximum and minimum diameter of grain (microns)
(iv) association of grain
(v) chemical composition of grain

The gold occurrences in the remaining thin sections were examined optically only. Visually estimated size and association were noted.

TABLE 1
LIST AND DESCRIPTIONS OF SAMPLES

RPC#			SAMPLE DESCRIPTION	TYPE OF ANALYSIS
BIRCHES ZONE				
7046	Hole MR117	57.5m	• pyrite in quartz/ankerite vein	O, AIA
7047		58.2m	• disseminated pyrite in illitized mafic volcanic	O, PROBE
7048		68.2m	• massive pyrite pods with quartz/ ankerite vein and illite	O
7049		68.9m	• disseminated pyrite in illite	O
7050	Hole MR70	29.0m	• massive pyrite with replacing specularite vein - with quartz and ankerite	O, PROBE
7051	Hole MR76	23.7m	• albitized rock with pyrite	O
7052		27.0m	• sheared, albitized mafic in contact with altered granite	
7053		24.2m	• pyrite vein associated with quartz and albite	O
7060	Hole MR28	43.8m	• massive pyrite and specularite	O, PROBE
7061		36.6m	• pyrite in sediments and carbonate vein	AIA
7062	Hole MR149	43.7m	• disseminated pyrite in quartz/ ankerite vein	
ROAD ZONE				
7056	Hole MR91	27.1m	• disseminated pyrite (~ 3%) in illite and barren quartz vein	O, PROBE
7057		28.4m	• disseminated pyrite in illite and quartz/ankerite vein	O
7058		29.0m	• nearly massive cubic pyrite in host and in quartz/ankerite vein	O
7059		29.2m	• disseminated pyrite in "layered" (illitized) sediments and quartz vein	O
7112	Sample from Ms. S. Watters		• gold in metasediments	O, AIA
PONDS ZONE				
7054	Hole MR28	16.4m	• disseminated pyrite in massive quartz/ankerite vein	O, PROBE
7055		14.0m	• disseminated pyrite in ortho-mylonitized granite	O
7089				O, AIA
7090	Grab samples from second bulk sample of MISPEC ore			O
7091				O
7092				O

O — Examination using optical microscope
AIA — Search for gold using automated image analyser
PROBE — Examination using electron microprobe

4.0 RESULTS

4.1 Birches Zone

4.1.1 Gangue Minerals

In the samples examined the mineral assemblages are very similar: quartz, illite, ankerite, pyrite, rutile, chalcopyrite +/- gold, albite, a potassium feldspar, calcite, muscovite, galena, hematite, magnetite, sphalerite, and various apparently secondary copper sulphides. These minerals exhibit two basic host rock textures which are as follows:

(i) Fine-grained illite matrix with from 15-50% medium to coarse-grained quartz +/- albite and potassium feldspar; medium-grained ankerite +/- calcite; and fine-grained irregular interstitial rutile which sometimes occurs as large skeletal blades or pseudomorphs.

(ii) Medium to coarse-grained granular albite, quartz (locally mozaic quartz), and ankerite; fine to medium-grained illite +/- muscovite; and minor fine-grained irregular rutile.

These textures are often cut by veins composed of medium to coarse-grained granular quartz and ankerite.

The pyrite exhibits very similar habits in all the samples. It occurs as medium to coarse-grained subhedral to anhedral poikiloblastic grains or aggregates (Figure 1). The outer rim of the pyrite tends to contain very few inclusions. The grains are also often fractured. The inclusions are composed of quartz, ankerite, illite, hematite, chalcopyrite, galena, and locally sphalerite. Rare fine-grained inclusion-free euhedral to subhedral grains are present. The coarser pyrite occurs in four different forms as follows:

FIGURE 1 Typical Irregular Coarse-grained Poikiloblastic Pyrite (P). Gangue(G), chalcopyrite (Cp), and gold (Au) inclusions are abundant along with fractures. (Sample 7050, backscattered electron image, bar is 100 microns).

(i) almost massive pods in quartz/ankerite veins
(ii) disseminated individual grains in host rock
(iii) almost massive vein with interstitial quartz, ankerite and minor illite
(iv) a concentration of individual grains and aggregates in an albite/quartz-rich illite host rock.

The finer-grained pyrite occurs in the host rock or quartz/ankerite veins adjacent to the higher concentrations of pyrite.

In one sample (7060) a massive pyrite vein overgrows an almost massive hematite vein (Figure 2). The hematite has partially altered to blocky magnetite.

In another sample various copper sulphides (bornite, covellite, digenite, chalcopyrite and carrollite), intergrown with each other in various combinations, occur as inclusions in the pyrite and interstitial to coarse-grained quartz and ankerite (Figures 3 and 4). Locally, in the same sample, a copper sulphate and dendritic copper/iron sulphate have overgrown the copper sulphides after the thin section was made (Figure 5a and b). Apparently the copper sulphides readily oxidize.

4.1.2 Gold Occurrences

Five of the samples examined contain gold in the form of native gold. They range in composition from 93% Au (7% Ag) to 99% Au (1% Ag) as indicated in Table 2. Eighty percent of these grains are 10-15 microns or smaller as indicated in Figure 6 and Table 3. Notice that the data in Figure 6 are image analysis measurements and those in Table 3 are based on visual estimates. In Figure 6 a cumulative size distribution curve based on surface areas of the grains is included with the curve based on frequency (number of grains). This curve is a good estimate of size versus volume %. It indicates that 80% by volume of the gold occurs as particles 20 microns in size or larger.

The gold grains occur in 9 different associations which are listed in Table 4. The most common occurrences are:

FIGURE 2 Pyrite (P) in a Hematite (H)/Magnetite (M) Concentration. Notice that the hematite appears to be partially altered to the blocky magnetite. (Sample 7060, backscattered electron image, bar is 100 microns).

FIGURE 3 A Mixture of Bornite (B), Covellite (Cv), and Carrollite (C). The mixture is interstitial to quartz gangue (G). (Sample 7050, backscattered electron image, bar is 10 microns).

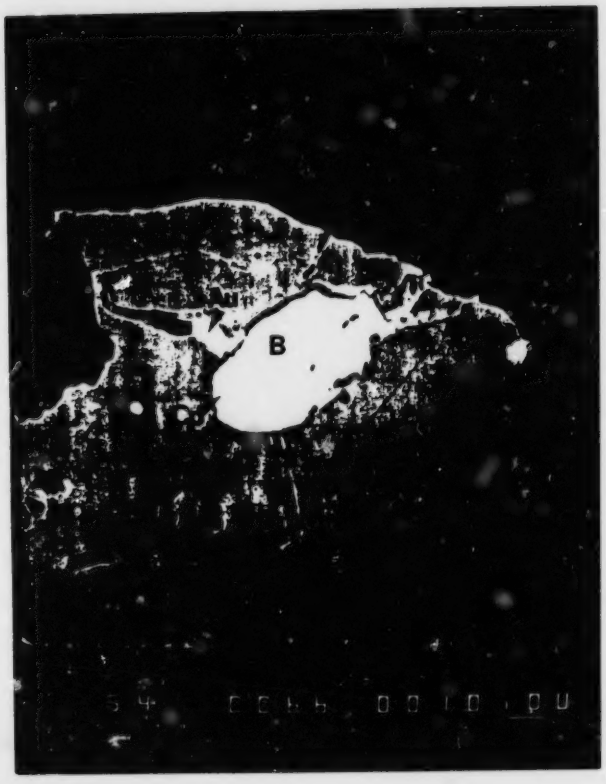
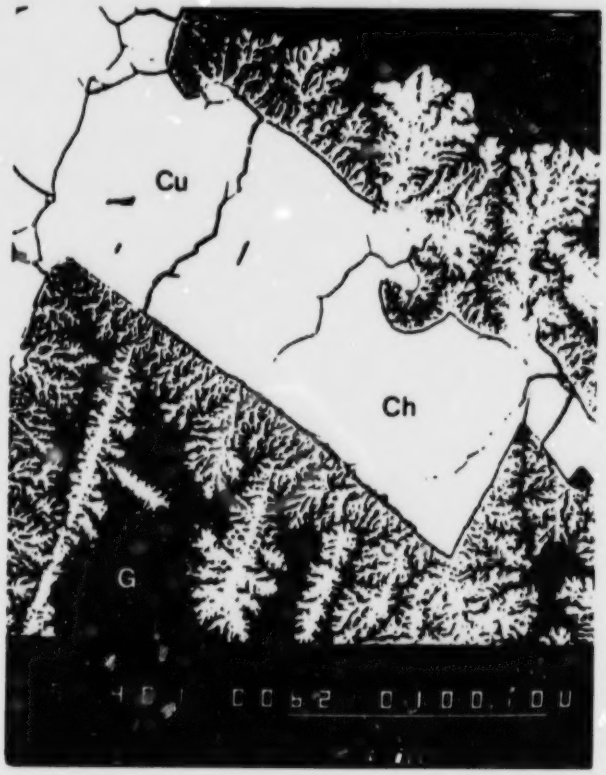


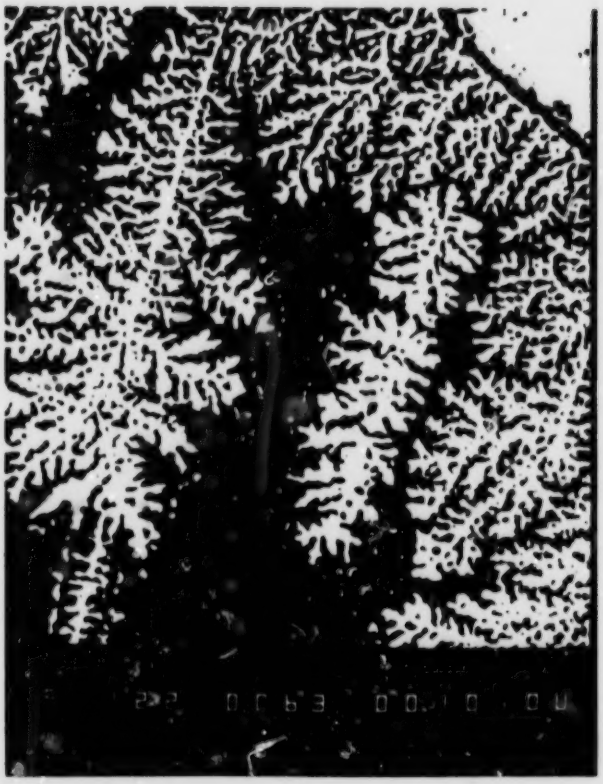
FIGURE 4 Bornite (B) and Chalcopyrite (Cp) in Pyrite (P). Notice the gold (Au) fracture filling. (Sample 7050, backscattered electron image, bar is 10 microns).

FIGURE 5 (a) Copper Sulphate (Cu) with Dendritic Copper/Iron Sulphate Overgrow Chalcocite (Ch) and Gangue (G) after Thin Section Preparation. (Sample 7050, backscattered electron image, bar is 100 microns).

(b) Detail of Copper/Iron Sulphate. (Sample 7050, backscattered electron image, bar is 10 microns).



(a)



(b)

TABLE 2

SEMI-QUANTITATIVE ANALYSES OF GOLD OBSERVED IN THE BIRCHES ZONE SAMPLES (%)

Sample: 7046

	1	2	3	4
Au	96	95	97	97
Ag	4	5	3	3

Sample: 7050

	1	2
Au	97	98
Ag	3	2

Sample: 7060

	1	2	3	4
Au	98	98	98	96
Ag	2	2	2	4

Sample: 7061

	1	2	3	4	5	6	7	8
Au	97	98	98	95	95	98	94	96
Ag	3	2	2	5	5	2	6	4

	9	10	11	12	13	14	15	16
Au	96	94	94	96	93	94	96	98
Ag	4	6	6	4	7	6	4	2

	17	18	19	20	21	22	23	24
Au	97	98	94	96	96	96	93	99
Ag	3	2	6	4	4	4	7	1

	25	26	27	28
Au	92	94	97	95
Ag	8	6	3	5

FIGURE 6
CUMULATIVE SIZE DISTRIBUTION OF GOLD OCCURRENCES
BIRCHES ZONE

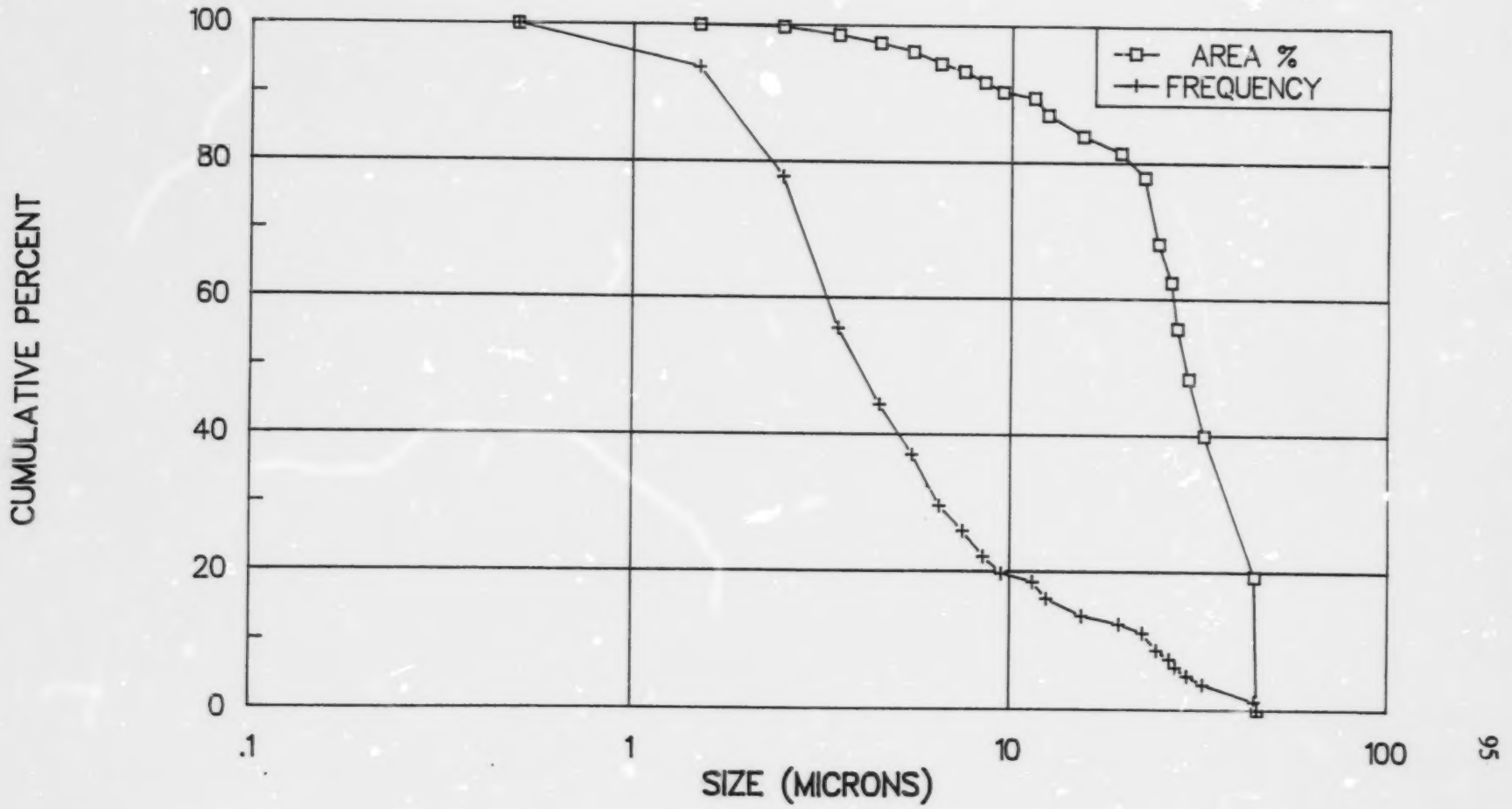


TABLE 3

VISUALLY ESTIMATED SIZE DISTRIBUTION OF GOLD PARTICLES IN SAMPLES 7043, 7050, AND 7060 OF THE BIRCHES ZONE.

Size Range (microns)	Number of Gold Grains	
1 -5	7	(27%)
5 - 10	8	(30.5%)
10 - 15	7	(27%)
15 - 20	3	(11.5%)
20 - 25	0	----
25 - 30	1	(4%)
30 - 35	0	----
35 - 40	0	----
40 - 45	0	----

TABLE 4

GOLD ASSOCIATIONS FOR THE BIRCHES ZONE SAMPLES

	Association	# of Individual Grains
(A)	Encapsulated in pyrite grain	28 (33%)
(B)	Encapsulated in pyrite grain with intergrowth of gangue	3 (4 %)
(C)	Between pyrite grains	10 (12%)
(D)	In fractures in pyrite grains	16 (19%)
(E)	On edge of pyrite grains	22 (26%)
(F)	Encapsulated in galena in ankerite vein	1 (1%)
(G)	Encapsulated in quartz grain near pyrite	1 (1%)
(H)	Encapsulated in Ca/Fe carbonate	1 (1%)
(I)	Interstitial to fine-grained illite	2 (2%)

(i) gold encapsulated in pyrite grain (33%)
(ii) gold on the edge of pyrite grains (26%)
(iii) gold in fractures in pyrite grains (19%)
(iv) gold between pyrite grains (12%)

In all cases the pyrite exhibits a coarse-grained (1-4mm in size) to massive habit as seen in Figure 7. In Figures 1 and 7 to 10 examples of five of the gold associations are illustrated.

As can be seen in the photomicrographs, the shapes of the gold particles vary. Both measured and visually estimated maximum and minimum diameters were also collected to determine a shape factor defined as dmax/dmin. A value of 1 indicates an equant grain and a larger value indicates elongation of the grains. As indicated in Table 5 almost 70% of the grains observed are roughly equant (shape factor of 1 to 2). Most of the remaining grains have a shape factor of less than 4 (28%). Very few (<5%) are long, thin fracture filling.

4.2 Road Zone

4.2.1 Gangue Minerals

Again, in the samples examined the mineral assemblages are very similar: illite, quartz, ankerite, a feldspar (either albite or a potassium feldspar), pyrite, and rutile. Goethite, lepidocrocite, and possibly hematite also occur in the highly oxidised samples (7056 to 7059). In sample 7112 pyrite is much more abundant and galena, sphalerite, chalcopyrite, and gold are also present.

In all five samples the host rock consists of highly folded and sheared fine-grained illite with, in sample 7112, small fine-grained lenses of quartz and minor potassium feldspar. Fine-grained irregular rutile (and hematite in the oxidized samples) occurs disseminated throughout. This rock is cut by veins composed of

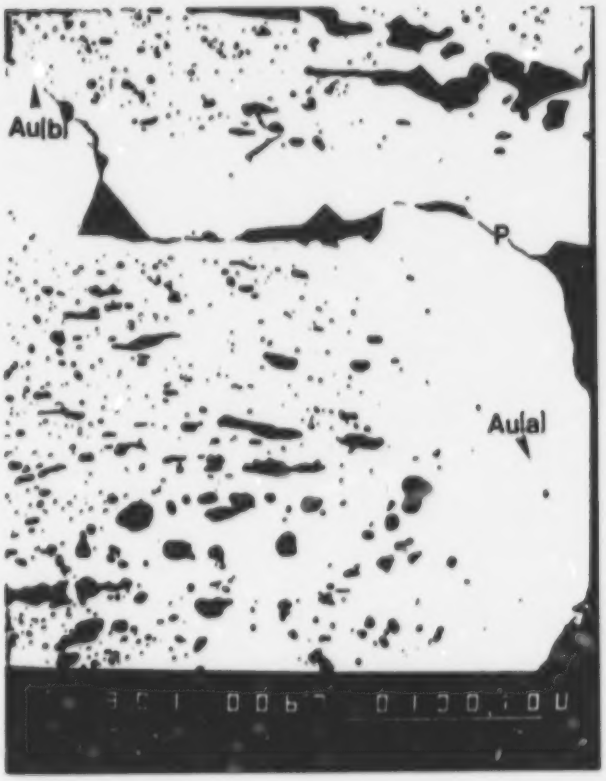


FIGURE 7 Native Gold (Au) Associated with Pyrite (P). "a" is encapsulated in pyrite and "b" is in between two pyrite grains. (Sample 7060, backscattered electron image, bar is 100 microns).

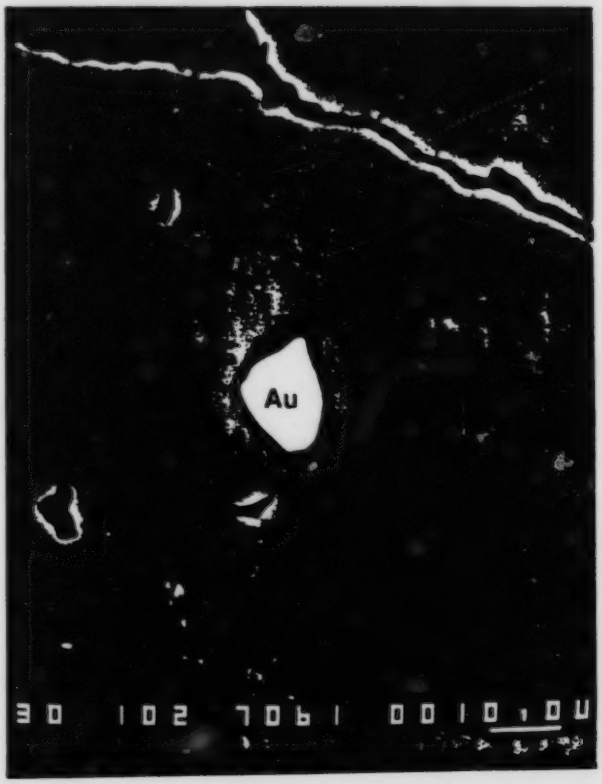


FIGURE 8 Three Examples of Native Gold (Au) Encapsulated in Pyrite (P). (Sample 7061, backscattered electron image, bar is 10 microns in both cases).

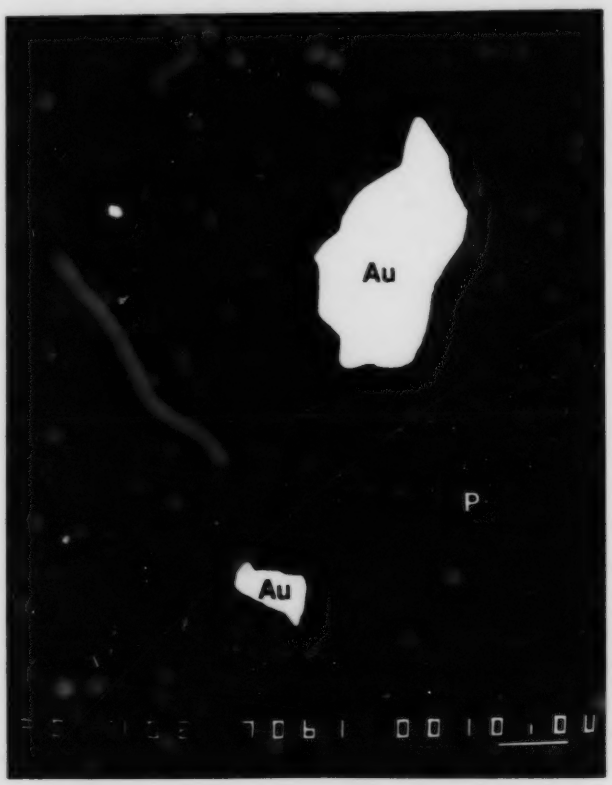
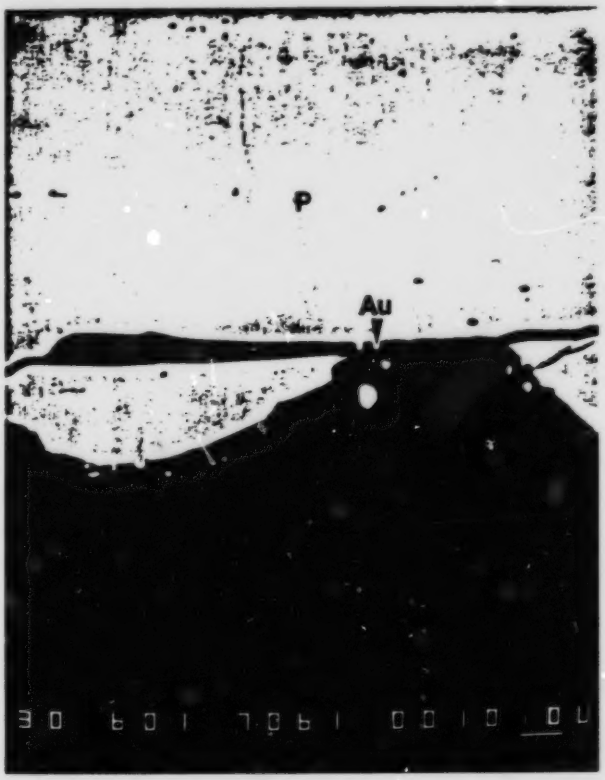
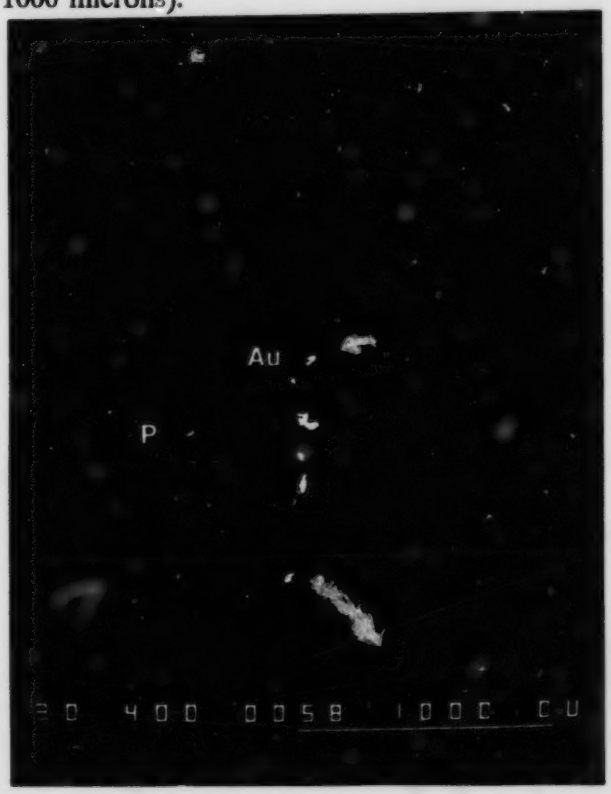


FIGURE 9 Example of Native Gold (Au) Fracture Filling in Pyrite (P). (Sample 7061, backscattered electron image, bar is 10 microns).



(a)

FIGURE 10 Two Examples of Native Gold (Au) on the Edge of Pyrite (P) Grains. (Sample 7061, backscattered electron image, (a) bar is 10 microns / (b) bar is 1000 microns).



(b)

TABLE 5

DISTRIBUTION OF SHAPE FACTORS FOR GOLD OCCURRENCES IN THE BIRCHES ZONE SAMPLES

SHAPE FACTOR	NUMBER OF GOLD GRAINS		
	Measured	Visually Estimated	Total
1 - 2	55	18	73 (68%)
2 - 3	17	4	21 (20%)
3 - 4	8	1	9 (8%)
4 - 5	1	0	1 (1%)
5 - 6	0	0	0 ---
6 - 7	0	0	0 ---
7 - 8	0	0	1 (1%)
8 - 9	0	0	1 (1%)
9 - 10	0	0	1 (1%)
>10	0	0	0 ---

medium to coarse-grained granular quartz, ankerite, and, in samples 7056 to 7059, albite. The veins cut the host rock subparallel to the shear zones.

The pyrite generally occurs as large subhedral poikiloblastic grains. Locally fine to medium-grained solid euhedral particles occur in the host rock. In the highly oxidized samples the pyrite has been almost completely altered to a mat of fine-grained goethite needles with minor lepidocrocite (Figure 11).

As stated above, galena, sphalerite, and chalcopyrite occur within the unoxidized sample (7112). Chalcopyrite is the least common, occurring as fine to medium-grained anhedral particles in the pyrite and the quartz/ankerite veins. Galena is present in much larger amounts and occurs as fine to medium-grained interstitial particles in the quartz/ankerite veins. It also occurs as inclusions in and coatings on pyrite and as "galena only" veins. Sphalerite is also common. It is situated in close proximity to the sphalerite/galena-rich gold-bearing vein (described in section 4.2.2). The sphalerite occurs as fine to medium-grained intersittial particles often intergrown with galena and pyrite.

4.2.2 Gold Occurrences

Gold, in the form of native gold and a gold-bearing telluride, was observed in sample 7112 only. It occurs in and around a massive sphalerite vein (50-95% of material) with fine to medium-grained interstitial galena particles. The vein thickness varies from 2 to 5 mm. Fine to medium-grained rounded anhedral pyrite, fine-grained chalcopyrite, Pb telluride, and Ag telluride are disseminated throughout. Inclusions of fine to medium-grained rounded quartz and ankerite commonly occur interstitial to the sphalerite. Illite is also present but is much less abundant. The vein cuts the host rock foliation and the barren quartz/ankerite veins at shallow angles.

FIGURE 11 Euhedral Pyrite (P) Altered to a Mat of Fine-grained Goethite (Go) Needles and Minor Lepidocrocite (L). (Sample 7056, backscattered electron image, bar is 100 microns).

A distribution of all the various associations is given in Table 6. Almost 75% of the native gold particles occur in the following associations:

(i) inclusions in ankerite (25%)
(ii) at the interface between sphalerite and ankerite (14%)
(iii) at the interface between galena, sphalerite, and ankerite (10%)
(iv) at the interface between galena and sphalerite (8%)
(v) inclusions in illite (7%)
(vi) at the interface between pyrite and illite (5%)
(vii) at the interface between galena and ankerite (4%)

The seven gold-bearing telluride grains occur as inclusions in illite (4 grains) and ankerite (2 grains), and at the interface between quartz and ankerite (1 grain). Approximately 65% of the native gold particles occur within the massive sphalerite/galena vein (Appendix A-2). Only one of the seven gold-bearing telluride particles occurs in this vein. Figure 12 shows an example of native gold at the edge of the sphalerite/galena vein.

The word "inclusions" has, in this case, been used in the descriptions of the associations to indicate grains surrounded by a single mineral. Because grain boundaries cannot be discerned, the gold-bearing grains are either encapsulated in the mineral or are at the interface between two grains of the same mineral.

The composition of the native gold particles ranges from 93 to 95% Au and 5 to 7% Ag (Table 7). The gold-bearing telluride particles are composed of from 12 to 26% Au, 40 to 48% Ag, and 32 to 34% Te (Table 7). The particles range in size from <1 to 58 microns with more than 80% of the particles smaller than 5 microns (Figure 13). Figure 13 also includes a size distribution curve based on surface area which estimates size distribution based on volume. According to this curve approximately 80% by volume of the particles are larger than 11 microns in size.

TABLE 6

GOLD ASSOCIATIONS FOR THE ROAD ZONE SAMPLE #7112

	Association	# of Individual Grains		
		Native Gold	Au-telluride	Total
(A)	encapsulated in Pb telluride	1	--	1 (0.4%)
(B)	inclusion in galena	13	--	13 (5.7%)
(C)	inclusion in sphalerite	1	--	1 (0.4%)
(D)	inclusion in ankerite	58	1	59 (25.7%)
(E)	inclusion in illite	17	4	21 (9.1%)
(F)	inclusion in quartz	8	--	8 (3.5%)
(G)	inclusion composed of native gold intergrown with Pb telluride in ankerite	1	--	1 (0.4%)
(H)	inclusion composed of native gold intergrown with gold-bearing telluride in ankerite	1	1	2 (0.9%)
(I)	interface between galena and sphalerite	19	--	19 (8.3%)
(J)	interface between galena and Pb telluride	1	--	1 (0.4%)
(K)	interface between galena and illite	3	--	3 (1.3%)
(L)	interface between galena and ankerite	10	--	10 (4.3%)
(M)	interface between galena and quartz	2	--	2 (0.9%)
(N)	interface between magnetite and ankerite	3	--	3 (1.3%)
(O)	interface between sphalerite and illite	4	--	4 (1.7%)
(P)	interface between sphalerite and ankerite	33	--	33 (14.3%)
(Q)	interface between quartz and ankerite	6	1	7 (3.0%)
(R)	interface between pyrite and illite	11	--	11 (4.8%)
(S)	intergrowth of native gold and Pb telluride on interface between pyrite and illite	2	--	2 (0.9%)
(T)	interface between galena, sphalerite and ankerite	22	--	22 (9.6%)
(U)	interface between galena, sphalerite and illite	1	--	1 (0.4%)
(V)	fracture - filling in pyrite (with galena)	3	--	3 (1.3%)
(W)	fracture - filling or veinlette in illite (with pyrite)	3	--	3 (1.3%)

FIGURE 12 Edge of Sphalerite/Galena Vein. The native gold (Au) occurs at the interface between sphalerite (S), galena (G) and ankerite (A). The small white dots are part of an x-ray map of gold. The sparse dots are background noise only. (Sample 7112, backscattered electron image, bar is 100 microns).

TABLE 7

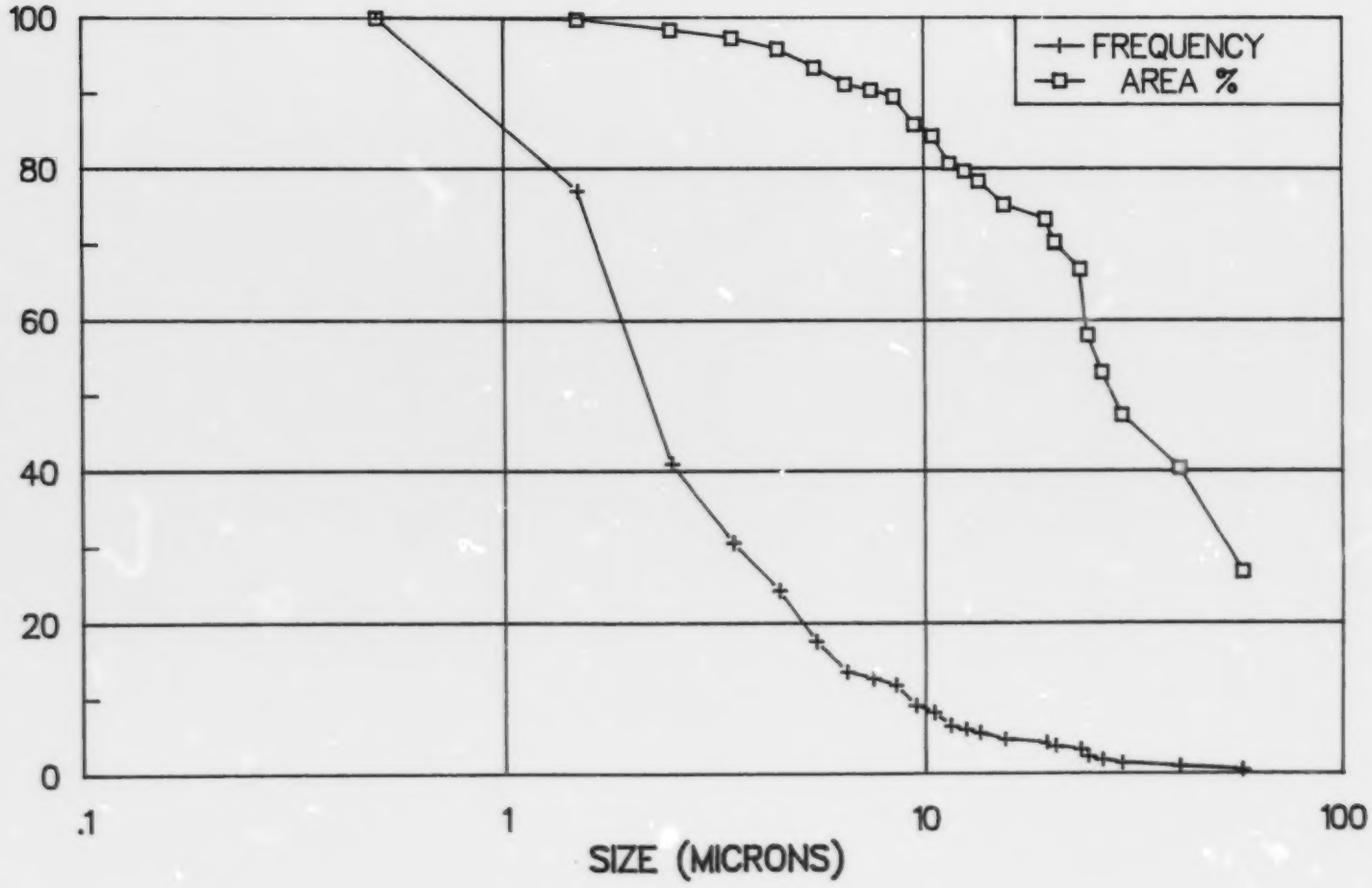
QUANTITATIVE ANALYSES OF GOLD OCCURRENCES IN THE ROAD ZONE SAMPLE # 7112 (%)*

Mineral	Gold*	Gold*	Telluride*	Telluride*
Au	94.85	93.14	26.10	25.40
Ag	5.87	6.76	40.93	40.07
S	0.01	0.08	0.06	0.07
Te	0.00	0.06	33.27	34.22
Pb	0.07	0.07	0.04	0.00
Fe	0.03	0.00	0.00	0.00
As	0.03	0.03	0.01	0.03
Zn	0.05	0.00	0.06	0.07
TOTAL	**100.91**	**100.14**	**100.47**	**99.86**

Mineral	Telluride	Telluride	Telluride	Telluride**
Au	25.05	23.49	22.96	12.75
Ag	41.59	42.49	43.62	48.00
S	0.00	0.00	0.00	0.00
Te	31.94	32.17	32.93	33.36
Pb	0.00	1.09	0.00	0.00
Fe	0.08	0.43	0.43	0.49
As	0.00	0.00	0.00	0.00
Sb	0.34	0.34	0.37	0.34
Cu	0.00	0.00	0.00	0.00
TOTAL	**99.01**	**100.01**	**100.31**	**94.35**

* From RPC Report # M/88/55
** Low total due to very small size of grain.

FIGURE 13
CUMULATIVE SIZE DISTRIBUTION OF GOLD OCCURRENCES ROAD ZONE



The shape of the gold-bearing grains varies as Table 8 indicates. The distribution of shape factors shown in this table is based on measured maximum and minimum diameters and defined as dmax/dmin. Approximately 60% of the grains are roughly equant (shape factor of 1-2), almost 35% are elongated (shape factor of 2-4), and less than 5% occur as lenses or veinlettes.

4.3 Pond Zone

4.3.1 Gangue Minerals

Only two samples from the pond zone were examined. They represented two distinctly different textures although both contained large, poikiloblastic pyrite. Sample 7054 is an example of a coarse-grained, granular quartz vein with poikiloblastic pyrite in its core. The pyrite contains quartz and hematite inclusions. Near this pyrite a narrow (10-100 microns) irregular vein of muscovite and fine-grained illite with minor fine-grained ankerite cuts the granular quartz. Fine-grained equant chalcopyrite grains occur within this vein. Fine to medium-grained irregular rutile occurs throughout the quartz vein but is most concentrated near the pyrite.

In sample 7055 the host rock is composed of granular fine to coarse-grained, strained quartz, albite, and possibly a potassium feldspar. Minor ankerite is also present. This rock has been highly illitized, the feldspars being the minerals most affected. Very fine to medium-grained illite (possibly with muscovite) occurs throughout. It exhibits a generally straight preferred orientation. Fine-grained, irregular particles of rutile occur throughout the illite, locally as skeletal tabular crystals, possibly pseudomorphs. Pressure shadows of both illite and quartz occur on the medium to coarse-grained anhedral to euhedral poikiloblastic pyrite. Locally fine-grained, semi-equant, rounded chalcopyrite particles occur in the gangue in small clumps.

TABLE 8

DISTRIBUTION OF SHAPE FACTORS FOR GOLD OCCURRENCES IN THE ROAD ZONE SAMPLE #7112

Shape Factor	Number of Gold Grains		
	Native Gold	Au Telluride	Total
1-2	135	4	138 (60.3%)
2-3	61	2	61 (26.6%)
3-4	17	1	17 (7.4%)
4-5	2	--	2 (0.9%)
5-6	1	--	1 (0.4%)
6-7	2	--	2 (0.9%)
7-8	3	--	3 (1.3%)
8-9	--	--	--
9-10	--	--	--
>10	1 (14-15)	--	1 (0.4%)

4.3.2 Gold Occurrences

No gold occurrences were observed in the Pond Zone samples.

4.4 Bulk Sample

4.4.1 Gangue Minerals

The four grab samples of bulk material are very similar. They are more oxidized than the core samples from the three zones. The host rock consists of medium to coarse-grained quartz that locally is highly strained. It exhibits minor subgrain development. Lenses and irregular, interstitial veinlettes of fine to medium-grained illite are common. The large lenses of fine-grained illite exhibit shearing texture or crenulation cleavage. Medium-grained, irregular, fractured, pyrite particles highly altered to goethite occur in these lenses. Fine-grained irregular rutile is also present. Overgrowing these textures is a fine-grained mixture of friable illite, quartz, iron oxide hydroxide, minor goethite and, possibly, minor carbonate.

4.4.2 Gold Occurrences

No gold occurrences were observed despite a detailed search using both optical and automated image analysis techniques (Table 1).

5.0 CONCLUSIONS

Gold-bearing minerals were observed in two of the four sample sets examined - the Birches Zone and the Road Zone. The occurrences in the samples from these zones are predominantly native gold although grains of gold-bearing telluride were observed in a Road Zone sample. The grains vary in size from <1 to 58 microns. In the Birches zone 80% by volume of gold is larger than 20 microns in size. Approximately 55% of the grains occur encapsulated in pyrite (in rare cases in quartz and ankerite also), or as fracture fillings in pyrite. To expose the gold grains the generally coarse-grained pyrite, quartz and, ankerite grains must be broken down. This would require a grind of 60 mesh (250 microns) or smaller. Since the gold grains are much smaller than 250 microns and pyrite does not readily break, a grind as fine as 400 mesh (38 microns) may be necessary to expose most of the gold occurring within the pyrite. The remainder of the gold (~45%) occurs at interfaces between various mineral grains. To expose the gold grains these mineral grains must be broken apart. Since most of the mineral grains are medium to coarse grained a grind of 60 mesh (250 microns) should be adequate.

The gold grains observed in the Road Zone sample set are 80% by volume larger than 11 microns in size. There are more smaller grains than in the Birches Zone sample set. Nevertheless, a large percentage of gold grains (~55%) occur at interfaces between various minerals. In addition, many of the gold inclusions occur in ankerite (~25%) which is a very soft mineral. Also, the gold grains that occur in other minerals may occur at grain interfaces within what are actually mineral aggregates. In general all these minerals are medium to coarse-grained, the medium-grained minerals tending to be in the coarser part of that size range. To expose the gold grains in these occurrences a grind of 140 to 60 mesh (106 to 250 microns) should be adequate since, in general, the mineral grains must be broken along grain boundaries but not across individual grains.

Total liberation of the gold grains in these two zones would require finer grinding. In addition, the shape of many of the gold grains in the Road Zone may cause problems even at a 400 mesh grind. Many of the particles are elongate and irregular. Also, they are

often intricately intergrown with sphalerite, galena, and/or gangue. Liberation of the main cores of the grains should not pose too many problems; however, the small protuberances on these grains will probably remain locked in gangue unless the material is ground very finely (to less than 10 microns).

A number of the minerals present in the samples examined should also be considered with regard to the leaching and recovery of the gold. Table 9 lists these potential "problem" minerals and their general abundances in the four sets of samples that were examined.

The copper and iron sulphides, present in all the sample sets, readily dissolve in the leachant. The resultig Cu and Fe ions interfere with the recovery of gold from the pregnant solution.

Soluble sulphates are a problem when thiourea is used as a leachant. Trace amounts of Cu and Cu/Fe sulphates were formed on a thin section from the Birches Zone sample set; however, it is unclear whether they are soluble and, therefore, a problem.
Carbonates are also a problem when thiourea is used as a leachant. Ankerite (CO_3) and minor calcite ($CaCO_3$) is present in significant amounts in all samples sets except the bulk sample.

Tellurides, found only in the Road Zone sample set, make variations in treatment necessary.

Illite will cause problems with percolation. It tends to break down into small grains that clog the pores and channels within the heap. Although it is not an expanding clay mineral, the clogging it may cause could be significant, especially since illite is a common mineral in all the sample sets examined.

TABLE 9

GENERAL ABUNDANCE OF "PROBLEM" MINERALS

	Minerals	Zones			Bulk
		Birches	Road	Pond	Sample
Cu:	cpy	m	m	m	--
	covelllite	tr	--	--	--
	digenite	tr	--	--	--
	bornite	tr	--	--	--
	carrollite	tr	--	--	--
Fe:	pyrite	c	c	c	m
	sphalerite	tr	c	--	--
Soluble (?) Cu/Fe Sulphates		tr	--	--	--
Ankerite/Calcite (carbonates)		c	c	m	tr
Tellurides (Au, Ag, PbP		--	m	--	--
Illite		c	c	c	c

c: common
m: minor
tr: trace
--: none

6.0 RECOMMENDATIONS

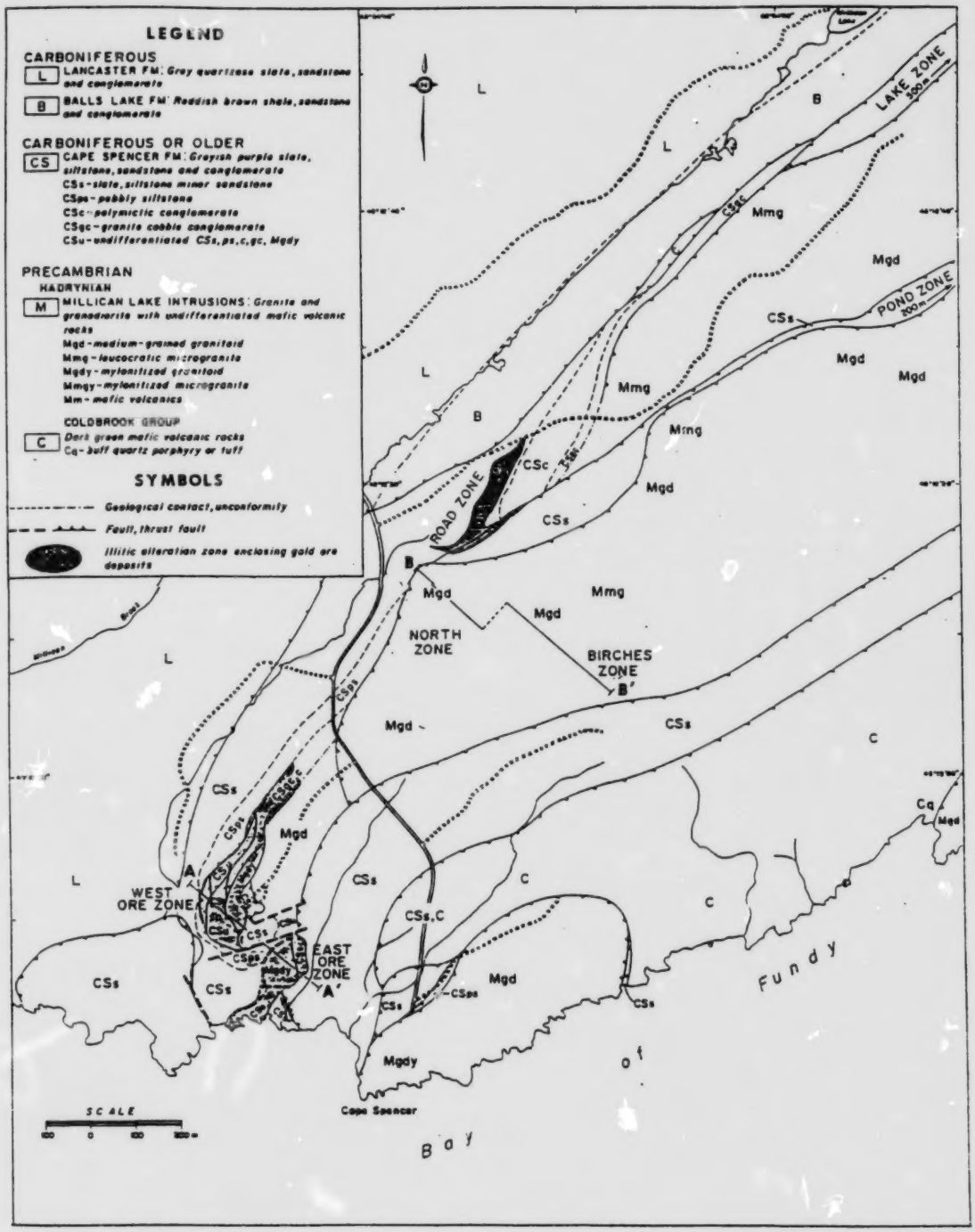
Examination of more gold-bearing samples is recommended because, although many gold-bearing grains were observed, the population studied is not large enough to give statistically sound results. Due to the reasonably coarse nature of the gold-bearing grains gravity concentration would be an excellent way to increase the amount of gold in a sample. Because relatively coarse grinding would be adequate, the associations of the gold could still be observed. In addition, the sample would be prepared from a large volume of material (as compared to the volume used in thin sections) and, therefore, would be more representative of the ore.

Testing a more representative bulk sample is also recommended. The current bulk sample is much more oxidized than any of the core samples examined for this report. Testing a sample collected by drilling or deeper trenching would give results that could be better applied to the whole deposit, not just the oxidized zone. Also, the mineralogical data in this report would be more relevant to this less oxidized bulk sample.

Finally, testing of various grinds on this unoxidized ore is recommended, taking into consideration the grinds proposed in the conclusions. Rocks and the minerals they contain can break in many different ways, not necessarily at grain boundaries first as is often assumed. Thus a coarser grind than those proposed in the conclusions may sufficiently expose the gold grains, or a finer grind may be necessary. The presence of fine-grained illite should also be considered when undertaking this testing.

APPENDIX A-1

Geological Map Showing the Locations of the Three Sample Zones. The bulk sample was collected from the Road Zone. The map is to be published in the 14th Annual Review of Activities for the New Brunswick Department of Natural Resources and Energy.


LEGEND
CARBONIFEROUS
L LANCASTER FM: Grey quartzose slate, sandstone and conglomerate
B BALLS LAKE FM: Reddish brown shale, sandstone and conglomerate
CARBONIFEROUS OR OLDER
CS CAPE SPENCER FM: Greyish purple slate, siltstone, sandstone and conglomerate
CSs - slate, siltstone minor sandstone
CSps - pebbly siltstone
CSc - polymictic conglomerate
CSgc - granite cobble conglomerate
CSu - undifferentiated CSs, ps, c, gc, Mgdy
PRECAMBRIAN
HADRYNIAN
M MILLICAN LAKE INTRUSIONS: Granite and granodiorite with undifferentiated mafic volcanic rocks
Mgd - medium-grained granitoid
Mmg - leucocratic microgranite
Mgdy - mylonitized granitoid
Mmgy - mylonitized microgranite
Mm - mafic volcanics
COLDBROOK GROUP
C Dark green mafic volcanic rocks
Cq - buff quartz porphyry or tuff
SYMBOLS
Geological contact, unconformity
Fault, thrust fault
Illitic alteration zone enclosing gold ore deposits
SCALE
LAKE ZONE
POND ZONE
ROAD ZONE
NORTH ZONE
BIRCHES ZONE
WEST ORE ZONE
EAST ORE ZONE
Cape Spencer
Bay of Fundy

APPENDIX A-2

SAMPLE DESCRIPTIONS

BIRCHES ZONE

Sample	Gold Occurrences	Host Rock	Metallic Minerals
7046	43 gold occurrences observed	illite quartz plagioclose ankerite chlorite tr apatite tr Host rock's (A) Host rock is fine-grained illite with ~ 10% lenses of medium-grained quartz +/- plagioclose, minor euhedral ankerite. (B) Medium to coarse-grained quartz, plagioclose and ankerite, generally equigranular overgrowing illite texture (A).	pyrite chalcopyrite rutile (A) fine to coarse-grained pyrite, anhedral to euhedral to almost massive. Large grains very poikiloblastic with inclusions of all other major phases. (B) Fine-grained anhedral chalcopyrite as inclusions in pyrite and ankerite and intergrown with rutile and illite. (C) Fine to medium-grained irregular often skeltal. Rutile skeletal grains may be pseudomorphs. Most common in illite-rich areas.
7047	No gold occurrences observed	illite quartz plagioclose (A) Host rock is composed of fine-grained unoriented illite with <15% medium-grained irregular quartz and plagiocolse, and euhedral to irregular poikiloblastic ankerite. (B) An irregular series of medium to coarse-grained granular quartz/ankerite veins cut the host rock. Quartz is strained and locally fractured.	pyrite rutile chalcopyrite (A) Disseminated medium-grained pyrite, euhedral with numerous inclusions giving it a poikiloblastic. testure. Overgrows rutile. (B) Fine to medium-grained generally skeletal rutile disseminated throughout but most common in illite-rich areas. (C) Fine-grained irregular chalcopyrite intergrown with rutile and also interskitial to large quartz grains.
7048	9 gold occurrences observed	illite quartz ankerite (A) Host rock of predominantly fine-grained illite showing no preferential orientation cut by medium to coarse-grained quartz/ankerite veins. (B) Pods of pyrite with interstitial ankerite, quartz and fine to medium-grained illite (+ muscovite) are associated with the veins. In both textures the quartz is not strained significantly.	pyrite galena chalcopyrite sphalerite tr rutile hematite (A) Two "pods" of aggregated anhedral pyrite grains often exhibiting fractures. Galena, chalcopyrite, gangue, hematite and sphalerite inclusions and fracture fillings are common. (B) Almost "vein-like" occurrences of galena progressing interstitially in ankerite veins and becoming coatings on and fracture fillings in pyrite "pods". Chalcopyrite intergrowths and gold inclusions occur locally. (C) Chalcopyrite occurs as small anhedral inclusions in pyrite and interstial to ankerite in veins.

7048 (cont'd)			(D) Sphalerite inclusions in pyrite occur but are rare. (E) Euhedral blades of hematite occur as inclusions in pyrite. (F) Fine-grained irregular rutile is common in illite-rich host rock.
7049	No gold occurrences observed	illite quartz ankerite muscovite (?)	pyrite chalcopyrite rutile
		(A) Host rock of very fine-grained illite exhibiting some preferential orientation which has been intricately folded. At hinges of folds and along host rock/vein interfaces the illite is medium-grained. Some muscovite may be present. Pressure shadows of illite/muscovite around pyrite are common. (B) Quartz/ankerite granular, medium to coarse-grained veins cut illite-(post folding). Minor illite is present between grains (intergranular).	(A) Generally rounded subhedral pyrite grains (0.25-4mm) disseminated throughout but less common in veins. Contain gangue and trace chalcopyrite inclusions that indicate rotation of pyrite grains during growth. Within veins often fractured (B) Chalcopyrite rare. Occurs as inclusions and fracture fillings in veins. (C) Small irregular rutile grains are common throughout illite.
7050	9 gold occurrences observed	illite quartz potassium feldspar ankerite	pyrite hematite chalcopyrite bornite covellite digenite carrollite unknown-green (Cu Sulphate ?) unknown-dendritic (Cu/Fe Sulphate ?)
		(A) Host rock is predominantly composed of fine-grained illite exhibiting no preferential orientation. Irregular lenses of quartz +/- feldspar (substantially altered) are present. Euhedral ankerite grains are also present scattered through illite. (B) Vein containing medium-grained granular quartz and ankerite with ~ 10% intergranular "influx" of illite (generally fine-grained but locally medium-grained). Some "mozaic" quartz is present locally as are small amounts of calcite.	(A) Generally medium to coarse-grained subhedral to anhedral pyrite often aggregated (almost massive). These grains are very poikiloblastic and fractured. Inclusions and fracture fillings of gangue minerals, hematite, chalcopyrite and the various other "secondary" minerals along with gold are common, disseminated. (B) Medium-grained blades of hematite-locally matted and folded. (C) Fine to medium-grained irregular (often rounded) inclusions in pyrite, and interstitial to quartz and ankerite. (D) Bornite +/- inclusions of carrollite, covellite, digenite and chalcopyrite in various combinations occur as inclusions in pyrite and interstitial to quartz and ankerite crystals. NOTE: gold appears to be spacially associated with these "secondary" ore minerals.

7050 (cont'd)			(E) The crystalline unknown green mineral and the dendritic unknown appear to be a third set of alteration minerals. They both overgrow bornite and covellite/digenite locally, but have formed after the thin section was made.
7051	No gold occurrences observed	quartz plagioclose (albite) illite (muscovite ?) ankerite (A) Medium to coarse-grained granular quartz and albite, locally overgrown by fine to medium-grained ankerite. Numerous ill-defined veinlettes or layers (lenses ?) of interstitial fine to medium-grained illite (and muscovite). The quartz and albite are strained. Along the illite/quartz-albite interface some coarse-grained illite (and possibly muscovite) occurs.	pyrite chalcopyrite sphalerite tr. rutile (A) Massive pods/lenses of irregular medium to coarse-grained pyrite, exhibit a poikiloblastic texture due to numerous inclusions of chalcopyrite, sphalerite and gangue. The pyrite is also often fractured. (B) Chalcopyrite and sphalerite occur as fine-grained irregular inclusions in pyrite. Chalcopyrite also occurs as very thin (1-2 μm) lenses in the quartz/albite/ankerite rock material. (C) Fine-grained irregular inclusions and interstitial grains of rutile are common,generally associated with illite.
7052	No gold occurrences observed	quartz albite illite ankerite (A) Very irregular fine to medium-grained albite and quartz (virtually unstrained) grains and clumps in a matrix of fine to medum-grained illite (and musvcovite ?). The albite is locally very altered and overgrowths of euhedral medium-grained carbonate (ankerite ?) are common. (B) Fine-grained illite with minor (<10%) irregular quartz grains and clumps (medium-grained and unstrained). Minor irregular medium-grained carbonate (ankerite ?) is also present. Coarse illite (muscovite ?) pressure shadows occur on the medium-grained pyrite. NOTE: These two rock types are in contact in this thin section.	pyrite chalcopyrite rutile (A) Predominantly medium to coarse-grained anhedral grains containing numerous gangue inclusions and fractures. Some rotation of pyrite is indicated by the inclusions. The pyrite occurs in rock type (B) and straddling the contact between the two rock types. Only very fine-grained irregular pyrite occurs elsewhere in rock type (A). (B) Fine-grained irregular chalcopyrite occurs in clumps (+/- similar pyrite) in rock (A). Local very thin (1-2 μm) long (25-75 μm) lenses of chalcopyrite also occur. The long dimension is approximately parallel to the contact between rock types. (C) Fine-grained irregular rutile occurs in both rock types but is more common in type (B) where it locally exhibits a preferred orientation parallel to the contact.

7053	No gold occurrences observed	quartz albite ankerite illite	pyrite chalcopyrite galena rutile
		(A) Generally medium to coarse-grained albite and quartz. The albite is significantly altered. Quartz is strained and locally is fine-grained, often exhibiting a mozaic texture. All this in a "matrix" of ~ 50-60% fine-grained illite. Medium-grained irregular ankerite is also present. (B) A medium to coarse-grained quartz, albite, and minor ankerite vein cuts through the host rock approximately parallel to the pyrite vein. Some illite is present interstitial to the other components. The quartz is strained and the albite shows some signs of alteration. (C) Within the pyrite vein medium-grained quartz, albite, ankerite, and fine-grained illite occurs as inclusions or interstitial to the pyrite.	(A) Within the pyrite vein medium to coarse-grained euhedral to massive pyrite with numerous fine to medium-grained gangue inclusions giving the pyrite a poikiloblastic texture. The pyrite is very fractured, often with chalcopyrite fillings. Elsewhere pyrite is rare, generally fine to medium-grained solid subhedral grains. (B) Chalcopyrite occurs as small inclusions and fracture fillings in the pyrite vein. Elsewhere it occurs as fine-grained irregular particles and thin (1-5 μm) long (25-100 μm) lenses. (C) Galena occurs in the pyrite vein only as fine to medium-grained rounded inclusions in the pyrite. (D) Fine-grained irregular rutile occurs throughout the thin section, associated generally with illite.
7060	8 gold occurrences observed	quartz ankerite illite	pyrite hematite magnetite chalcopyrite galena
		The quartz, ankerite and illite occur as inclusions in pyrite and interstitial to both the pyrite and hematite/magnetite. This interstitial material is a mixture of medium to coarse-grained strained quartz, fine-grained illite, and medium-grained ankerite. Quartz is the most common of the three. Within the hematite/magnetite-rich zone both quartz and the ankerite are finer grained than in the pyrite-rich zone. Moziac quarts is also more common.	(A) Pyrite in pyrite-rich zone is coarse-grained, euhedral to massive clumps, and contains numerous inclusions of gangue, chalcopyrite, galena, hematite, and gold. On outer edges inclusions are less numerous. In hematite/magnetite-rich zone individual pyrite grains resembling those elsewhere in the thin section do occur and contain locally large, often bent, hematite blades. (B) Coarse-grained blades and aggregates of fine-grained blades of hematite exhibit extreme folding in the hematite/magnetite zone. Elsewhere hematite occurs as fine-grained euhedral blades in pyrite. (C) Magnetite occurs as blocky grains in aggregates which exhibit the same blake-like form as the hematite. These "blades" often contain hematite cores.? altered hematite? (D) Chalcopyrite and galena occur as fine to medium-grained inclusions in the massive pyrite. Chalcopyrite also occurs as fracture fillings in the pyrite.

ROAD ZONE

Sample	Gold Occurrences	Host Rock	Metallic Minerals
7056 to 7059	No gold occurrences observed	illite quart ankerite albite (A) Highly folded fine-grained illite with many semi-parallel shear zones evident. (B) Quartz +/- ankerite and albite veins cut through the illite generally semi-parallel to the shear zones, however, they are very irregular and do locally cut across this lineation. The 3 common minerals exhibit a granular texture and are generally medium to coarse-grained. NOTE: all samples extremely oxidized!	pyrite goethite lepidocrocite rutile Fe-oxide (hematite ?) (A) Some pyrite is present. Although now highly altered it was once medium to coarse-grained and euhedral to subhedral. The pyrite predominantly occurs in the fine-grained illite. (B) The pyrite has been altered to a fine mat of goethite needles with minor lepidocrocite (probably, definitely Fe oxide) along fractures and on the outside of the pseudo-morph (Figure 12). (C) Within the illite fine-grained irregular rutile and some Fe oxide (probably hematite) is disseminated
7112	230 gold occurrences observed NOTE: all the occurrences of gold occur in and around a sphalerite/galena vein cutting shallowly across the foliation in the host rock. Because of the complicated nature of the gold occurrences details are given in Appendix A-2.	illite quartz potassium-feldspar ankerite (A) The host rock is predominantly fine to medium-grained illite exhibiting preferential lineation with shear zones very evident. Numerous very fine-grained lenses of quartz +/- (possibly) potassium feldspar are present between the illite with their long directions parallel to the shear zones. (B) Illite host rock is cut at a shallow angle by anastomosing veinlettes composed of fine to coarse-grained granular quartz and ankerite. The quartz is very strained. Note that these veinlettes are themselves cut at a shallow angle by the gold-bearing vein.	pyrite galena sphalerite (?) chalcopyrite rutile Pb telluride Ag telluride (A) Fine to coarse-grained pyrite is disseminated throughout the sample but is very rare in the quartz/ankerite veins. The larger grains are poikiloblastic (mainly quartz, illite, and ankerite inclusions) and fractured. (B) Galena is quite common and occurs interstitial to the quartz and ankerite, and as inclusions in and coatings on pyrite. "Galena only" veins are also common. (C) Chalcopyrite is not common, mainly occurring as fine to medium-grained inclusions in pyrite and irregular concentrations in the quartz/ankerite veins. (D) Sphalerite is less common than galena, mainly situated near the gold-bearing vein as interstitial grains often intergrown with galena, and pyrite. (E) Fine-grained irregular rutile is disseminated throughout the illite-rich host rock.

7112 (cont'd)

(F) Fine-grained Pb and Ag tellurides occur in the massive sphalerite/galena vein near the gold, locally intergrown with gold or gold-bearing telluride.

Gold Bearing Vein: A massive sphalerite (50-95% of vein)/galena (interstitial)vein with fine to medium-grained rounded anhedral pyrite disseminated throughout and rare fine-grained chalcopyrite. The vein is 2 to 5 mm wide. The gold occurs in long irregular grains often intergrown with galena and locally Au-telluride. Fine-grained Pb telluride and Ag telluride often occur near the gold (locally intergrown).
Inclusions of fine to medium-grained, rounded quartz and ankerite commonly occur interstitial to the sphalerite. Illite is much less common and appears to be overgrown.

POND ZONE

Sample	Gold Occurrences	Host Rock	Metallic Minerals
7054	No gold occurrences observed	quartz illite/muscovite ankerite	pyrite hematite rutile chalcopyrite
		(A) "Host rock" coarse-grained granular quartz vein. (B) Near a concentration of pyrite a narrow (10-100 microns wide) irregular vein of muscovite and illite (fine-grained) with minor fine-grained ankerite cuts the granular quartz.	(A) Local subhedral to anhedral pyrite, 1 to 5 mm in size, occurs in core of the quartz vein. The pyrite contains irregular quartz and (rarer) hematite inclusions. (B) Fine to medium-grained irregular rutile occurs throughout the vein but most concentrated near the pyrite. (C) Fine-grained equant chalcopyrite grains occur in the illite/muscovite vein and nearby in the massive quartz.
7055	No gold occurrences observed	quartz albite potassium feldspar (?) illite (+ muscovite ?) ankerite	pyrite chalcopyrite rutile
		Host rock is comprised of granular-fine to coarse-grained quartz, albite and possibly a potassium feldspar. The quartz is strained. The feldspars are highly altered to illite. The rock has been illitized. Very fine to medium-grained illite exhibits a straight preferred orientation throughout. Pressure shadows of both illite (muscovite ?) and quartz occur on pyrite grains. Ankerite is also present but not common.	(A) Anhedral to euhedral, medium to coarse-grained pyrite occurs disseminated throughout the sample. The pyrite is poikiloblastic to almost skeletal due to blebby inclusions along the cube surfaces in the particles. Locally these inclusions of the host rock minerals are reotated. Locally inclusions of rutile also occur. Apparently they have been overgrown by the pyrite. Their habit is the same as elsewhere in the sample. (B) Rare rounded, semi-equant, fine-grained particles of chalcopyrite occur in the gangue in small clumps. (C) Fine-grained irregular particles of rutile occur generally within the illite. Some occur as skeletal tabular grains-possibly pseudomorphs.

APPENDIX A-3

DETAILS OF GOLD OCCURRENCES IN SAMPLE #7112 FROM THE ROAD ZONE

OUTSIDE SPHALERITE/GALENA VEIN

	ASSOCIATION	# OF INDIVIDUAL GRAINS NATIVE GOLD	AuAg TELLURIDE
(A)	inclusion in illite	17	4
(B)	inclusion in ankerite	27	1
(C)	inclusion in ankerite composed of native gold intergrown with Pb telluride	1	0
(D)	inclusion in ankerite composed of native gold intergrown with AuAg telluride and Pb telluride	1	1
(E)	interface between interstitial galena and illite	1	0
(F)	interface between quartz and ankerite	5	0
(G)	interface between sphalerite and ankerite	2	0
(H)	interface between pyrite and illite	11	0
(I)	intergrowth of Pb telluride and native gold on interface between pyrite and illite	2	0
(J)	fracture-filling in pyrite with galena	3	0
(K)	fracture-filling or veinlette in illite with pyrite	3	0
	TOTAL	73	6

IN SPHALERITE/GALENA VEIN

	ASSOCIATION	# OF INDIVIDUAL GRAINS NATIVE GOLD	AuAg TELLURIDE
(A)	encapsulated in Pb telluride	1	--
(B)	inclusion in galena	13	--
(C)	inclusion in sphalerite	1	--
(D)	inclusion in ankerite	31	--
(E)	inclusion in quartz	8	--
(F)	interface between galena and sphalerite	19	--
(G)	interface between galena and Pb telluride	1	--
(H)	interface between galena and ankerite	10	--
(I)	interface between galena and illite	2	--
(J)	interface between galena and quartz	2	--
(K)	interface between sphalerite and ankerite	31	--
(L)	interface between sphalerite and illite	4	--
(M)	interface between magnetite and ankerite	3	--
(N)	interface between quartz and ankertie	1	1
(O)	interface between galena, sphalerite and ankerite	22	--
(P)	interface between galena, sphalerite and illite	1	--
	TOTAL	150	1

AUTHORIZATION

DEPARTMENT: **MINERALS AND FUEL TECHNOLOGY**

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APPENDIX 2

RECOVERY OF GOLD FROM HEAP LEACH SOLUTION

A REPORT BY THIBAULT AND ASSOCIATES INC.

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RECOVERY OF GOLD FROM HEAP LEACH SOLUTION

Preliminary Assessment of Technologies

April 1990

Prepared for:

Research and Productivity Council

In Association With:

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DISCLAIMER

An *independent consulting firm* has performed a review of gold recovery technologies based on published literature. The technology has been tested on *Cape Spencer occurrences*, acknowledging the developers of the technology and/or previous studies done on the ore. The conclusion of this study relies on the accuracy of published literature and data derived from standard industrial testing procedures.

This study is to be used only for the purpose of assisting with the development of Cape Spencer occurrences. Technological assessment and testing done under this study are not intended for commercial design or ownership of technology for commercial benefit.

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1.0 EXECUTIVE SUMMARY

Cost comparisons on two different extraction plant methods of gold recovery from heap leach solution are presented. Technical implications of solution impurities on technology selection was reviewed and an evaluation format was developed to assess the relative maturity of available technologies. The assessment revealed zinc precipitation and carbon-in-column to be the two most viable technologies to treat Cape Spencer field occurrences. Plant designs for each technology are based on initial laboratory testwork supplemented by information available in the literature. Conceptual capital and production costs are interpolated from generally accepted plant practice substantiated by a literature review of current operating plant data. Based on 250,000 tonnes/year of ore, the capital and production costs for either technology was approximately equal, indicating the need for a rigidly monitored pilot plant study of both methods prior to a final technology selection.

2.0 INTRODUCTION

During the past decade, heap leach has emerged as a viable technology for the commercial development of various precious metal occurrences in New Brunswick. Heap leach and agglomeration methods were initially developed on the former Caribou Gossan by Anaconda in 1983. Heap leach was also employed by Gordex Mineral Limited in 1985 for the production of gold from the siliceous gold occurrences of Cape Spencer. Both operations originally used zinc precipitation to recover gold from cyanide solutions but the Gordex operation converted to carbon-in-column technology, prior to ceasing operations in 1987. Zinc precipitation was also used by Heath Steel Mines in 1984 employing in-tank leaching to treat Gossan and, most recently, by Nova Gold Resources Inc. at the Murray Brook enclosed heap leach operation.

The selection of an optimum technology to recover precious metals from cyanide solutions is an integral part of a development strategy to commercialize precious metal deposits. Recovery from cyanide leach solutions is dependent on solution composition and volume, which is ore specific but may vary within a precious metal deposit. Identifying the most viable technology for gold recovery requires an assessment of commercial and developing technologies, as well as a development test program to characterize each technology on specific heap leach solutions. This report presents a number of technical and economic factors which should be considered in planning, testing and designing unit operations for gold recovery from cyanide solutions.

2.1 Study Objective

This study was completed as an integral part of a project entitled *Processing Options for Gold Recovery of Cape Spencer Ores* originated by the New Brunswick Department of Natural Resources and Energy as contracted to the Research and Productivity Council. Under subcontract to the Research and Productivity Council (RPC), the objective of this study was to determine the

most applicable technology for gold recovery from Cape Spencer heap leach solutions based on a literature review, test results and a comparative economic assessment.

2.2 Scope of Work

The work included under this study is as follows:

1) Development of evaluation criteria for assessing both existing and developing technologies used to recover gold from cyanide (pregnant leach) solutions. Assessment of technologies is based on a literature search completed by RPC.

2) Assessment of gold extraction plant capacity based on pilot column leach data and an annual commercial production of 250,000 tonnes of ore.

3) Conceptual design of extraction plant flowsheets and size equipment for the two most applicable technologies. Battery limits for the flowsheet includes pregnant solution feed, barren solution effluent and unrefined gold precipitate or cathodes.

4) Determine order of magnitude capital and operating costs, within a 30% range, for technologies sized in item 3.

2.3 Background

A total of five commercial or emerging technologies by which gold may be recovered from heap leach - cyanide solutions were reviewed in this study. A description of each technology's operating principle is included in the test program report by RPC. Zinc precipitation and carbon-in-column represent the most frequently used technologies by today's mining industry. Some alternative technologies with commercial possibilities include resin ion

exchange, solvent extraction and direct electrowinning. The most advanced of these is resin ion-exchange in continuous operating columns.

The choice of one technology over the other results from detailed consideration of a large set of evaluation criteria for each project. Since ore types are very diverse in composition, the key parameters for selection of a gold recovery technology should be based on pregnant solution compositions and characterization of gold extraction by continuous operating test programs. The two most relevant factors which determine the technical and economic efficiency of the subject technologies are as follows:

1) ***Selectivity and Recovery***: The optimum technology must be selective to the recovery of gold, which is directly related to the purity of doré and the consumption of fluxing agents used in smelting precipitates and/or electrowon steel wool. In the case of highly fouled solutions (high concentrations of undesirable metals which generally reduce the purity of doré), the gold recovery technology must be selective and at the same time have a low dissolved loss. (Dissolved loss refers to the amount of gold in solution lost to tailings).

2) ***Reaction Rates***: The process of extracting gold from the pregnant solution must be rapid to prevent excess operating and capital costs. For the same reason, gold stripping from the extraction media must proceed quickly.

In conjunction with gold recovery efficiency, the heap operating philosophy or "solution management" impacts directly on the overall efficiency of the heap leach operation. An integral part of solution management is the "solution application flux", known as the volumetric flow rate of barren solution to the heap per unit area of ore. A reduction in solution application flux will result in an increase in the pregnant solution gold content and will also result in a down-sizing of the extraction plant. Based on heap leach operating experience, the same weight of gold extracted per day can be obtained by lowering barren solution flow and increasing pregnant

solution grade, for solution application fluxes in the range of 2.5 to 11.0 lt/hr/m^2. Since flowrate reduction of pregnant/barren solution can cut operating and capital costs, an assessment of the ideal solution flowrate should be included in the technology selection criteria.

Chemical control is also part of solution management and is directly related to the ore and solution composition. Controlling the chemical composition of a pregnant solution is difficult because of the relatively long solution residence time in the heap. Chemical control is generally determined by the operation of the extraction plant and daily chemical addition is as effective as continuous chemical addition. A number of factors effecting chemical control include:

- cyanide concentration and cyanide-gold/cyanide-metal ratios.
- dissolved oxygen
- alkalinity and carbonates
- fouling metals such as copper, cobalt, nickel etc.
- turbidity and scale
- thiocyanide and ferrocyanide

Assessment factors for technology selection have not been defined for the Cape Spencer type occurrences. The Cape Spencer group of occurrences has two distinct zones with respect to heap leach processing of ores. The surface or oxidized zone contains gold associated with iron oxyhydroxide and, to a lesser extent, gold encapsulated in pyrite. In the sub-surface zone, gold is predominantly associated with pyrite. Preliminary testing on the sub-surface zone by the Department of Natural Resources and Energy (DNRE), in 1986, indicated the presence of fouling metals in pregnant solution and problems with solution percolation in oxidized ores. A detailed assessment of the gold recovery from solution was not completed by DNRE and is the subject of this project.

In order to address the processing problems posed by fouling metals, a review and assessment of technologies available in published literature is included in Section 3.0 and Section 4.0. The evaluation criteria used to assess the two most probable technologies, based on design and costing of the flowsheets, is included in Section 5.0 and 6.0. To quantify the amount of pregnant solution to be fed to the extraction plant, a water balance spreadsheet program was also developed.

3.0 LITERATURE REVIEW AND TECHNOLOGY EVALUATION

The project scope includes the investigation and selection of an optimum technology for the recovery of gold from heap leach solutions with particular emphasis on solution fouling and impurity levels. It is recognized that selection of the ideal gold recovery technology is dependant on pregnant solution composition and that optimization of each technology is also solution specific. Therefore, discussion in this section is aimed at providing general criteria for pregnant solution characterization. A literature review of the technologies was based on the following objectives:

a) Technical and economic implications of pregnant solution composition on the efficiency of each technology. Methods of chemical control are reviewed.

b) Development status of each technology and the implications for commercial use.

The information provided in this review was used to assess the technologies using the evaluation criteria and is discussed in Section 3.5.

3.1 Zinc Precipitation

Zinc precipitation (Merrill Crowe) is the most widely used technology for commercial operations, providing the pregnant solution is suitable for optimum gold recovery. Ores with high silver values use zinc precipitation because of high costs of silver stripping in carbon-in-column system. Zinc precipitation is relatively easy to operate and few personnel are required for a continuous operation. Plants having a wide variation in gold content generally select zinc precipitation because of the negligible affect on the operating cost.

To maintain an efficient use of zinc powder, zinc precipitation requires a relatively high concentration of gold in the pregnant solution. The consumption of zinc also depends on the presence of other metals in the pregnant solution. Average consumption of zinc is 1 to 2 grams per gram of gold and silver. The addition of lead nitrate (1 gram of lead nitrate per 10 g of zinc) will increase the recovery of gold and silver. Aluminum can be used as an alternative to zinc but can cause calcium aluminate fouling of filters and difficulties in smelting.

3.1.1 Silver/Gold

Zinc precipitation is most effective for high silver to gold ratios in the range of 5:1 to 20:1. Both silver and gold are reduced to native states and the cyanide complexes with zinc.

Loss of gold in the zinc precipitation circuit is generally due to solid separation. Barren solution grades of 0.01 g Au/t are generally obtained resulting in 97 - 98.5% recovery.

The stoichiometric amount of zinc added should be based on pregnant solution testing.

3.1.2 Sulfide

Concentrations of sulfide ion in the range of 10 to 15 ppm sulfide will inhibit zinc precipitation. This is believed to be due to the protective coating of zinc sulfide on zinc. Therefore, as the sulfide concentration increases, the consumption of zinc increases proportionally.

3.1.3 Alkalinity/Cyanide

A pH of 10 or greater is required to precipitate precious metals and minimize calcium precipitation. Certain concentrations of copper will also result in high consumption of zinc and will affect the efficiency of precipitation. The pregnant solution should have a high concentration of cyanide to eliminate the effects of copper precipitation.

Free cyanide ions promote the anodic dissolution of zinc but an excess of cyanide will result in the evolution of hydrogen. Anodic current densities are generally sustained around 10-13 mA per cubic cm and free cyanide and calcium oxide concentrations are maintained around 150 ppm and 100 ppm respectively. Free cyanide must be present in the pregnant solution to dissolve zinc.

3.1.4 Arsenic and Antimony

Arsenic and antimony in the pregnant solution have a negative effect on the precipitation kinetics. Concentrations of about 1 ppm will reduce the precipitation rate by 20%. In some cases, sodium sulfide will be required to precipitate arsenic and antimony as sulfides. However, silver may be lost with precipitate, if this is done.

3.1.5 Oxygen

Pregnant solutions average 7 to 8 ppm oxygen and the deareation of the solutions should reduce oxygen to levels less than 1 ppm. Traces of oxygen will have a deleterious affect on precipitation because of passivation of the zinc surface. Temperature has an effect on the efficiency of deareation because of the increased solubility of oxygen in water at lower temperatures. Therefore, solution heating may become a consideration in the winter operation of zinc precipitation circuits.

3.1.6 Suspended Solids

The efficiency of zinc precipitation is directly related to the clarification of pregnant leach solutions. Pre-coat pressure filtration is known to provide the best operational results. High dissolution losses are due to incomplete washing in the clarification of the pregnant leach solutions.

Clarification is generally quite successful in lowering pregnant solutions from 400 ppm solids to less than 5 ppm solids. Optimum clarification of pregnant solution is done by proper selection of filter cloth and pre-coat.

3.2 Carbon-In-Column

During the last decade, carbon adsorption technology has gained a wide acceptance in the mining industry and, most recently, carbon-in-column (CIC) systems have been commercially installed. CIC is less sensitive than zinc precipitation to total suspended solids in feedstocks which may improve metal recovery.

The cyanide to gold ratio on the carbon is generally 2:1 to 1:1. This is due to the precipitation of AuCN and the reduction of the gold cyanide complex to elemental gold by carbon. Similarly, as was the case with zinc precipitation, the metal cyanides formed in the leach solution will consume cyanide. Therefore, the consumption of cyanide is not directly related to gold content.

The loading of fouling agents is dependent on the leachability of the various minerals in the ore. Some of the most common fouling agents in the pregnant solution include copper, calcium, and silica. The degree of carbon fouling is also related to the efficiency of carbon stripping.

Various options are available for stripping the gold from the carbon. Acid treatment of the carbon will efficiently remove iron, zinc and nickel but the removal of copper may be to a lesser extent. High carbonate solutions limit gold adsorption and present difficulties in carbon stripping. The most common methods for stripping include a caustic/cyanide solution at 100° C to 150° C for 40 to 70 hours. The time required to complete stripping is a limitation to

carbon-in-column technology. Most of the capital requirements for carbon adsorption is due to elution (stripping) and regeneration of carbon in rotary kilns.

3.2.1 Calcium

High calcium concentrations in the pregnant solutions will consume reagents used in the stripping circuit. Calcium fouling of the carbon and adsorption of gold on the carbon is related to the ion paired species $Ca(Au(CN)_2)_2$. The high ionic strength of solutions containing calcium will assist in the gold adsorption. Adversely, the formation of calcium carbonate has a negative affect on gold adsorption. Calcium loading of 10-20 Kg/t has a marginal affect and loadings around 50 Kg/t result in a decline of adsorption rates.

Carbon is easily fouled with calcium as well as magnesium carbonates which may be removed by extensive washing and by calcination. Sulfuric acid is also used to adjust pH of pregnant solutions and dissolve carbonate.

3.2.2 Silver

Silver is not readily recovered at low cyanide concentrations and carbon has a poor adsorptivity for silver. In a carbon adsorption flowsheet, sodium sulfide is generally used to precipitate silver as silver sulfide. If silver levels are higher than gold in the pregnant solution, then zinc precipitation is more cost effective.

3.2.3 Sulfide

High levels of sulfide ion in the pregnant solution will interfere with cyanidation and/or recovery of gold from solution. A level of 320 ppm of sulfide concentration is considered high.

3.2.4 Organics

Organic foulants are of major concern in carbon circuits. Silica, iron oxides and carbonates may be removed by routine acid wash and are monitored - by thermogravimetric

analysis - to determine the efficiency of elution. In the case of heap leaching, organic foulants will not be a limitation.

3.2.5 Alkalinity

Alkalinity has an effect on the loading of Au, Cu, Ca and SiO_2. A decrease in the pH of the pregnant solution from 10.5 to 8.0 will increase loading of gold and copper, but will, however, decrease the loading of silver, calcium and silica. At a pH of 8, there is a greater destruction of cyanide.

3.2.6 Copper

High copper loadings may result from the use of low cyanide concentrations. At low cyanide concentrations, copper and other base metals may compete for the active sites on the carbon. Copper loads on carbon in the ratio of 1:5 of CN to Cu and in order to avoid excess loading of copper, ratios of 3:2 and 2:3 are used. The reason for this has been related to the formation of $Cu(CN)_2^-$, which adsorbs well on carbon. Species carrying a negative 2 or negative 3 charge do not load appreciably on carbon.

3.2.7 Iron

Iron is not known to affect gold loading rates on carbon.

3.2.8 Cyanide

In pregnant solutions, the free cyanide should be controlled in the range of 100 to 150 ppm at a pH of 10.3. High cyanide concentrations around 11,000 ppm CN will inhibit loading rates. Loading rates are not effected at 2,400 ppm CN. The molar ratio of cyanide to gold should be approximately 2:1.

3.2.9 Thiocyanide

Not as detrimental as copper and silver but will decrease the loading rate.

3.2.10 Oxygen

Low oxygen levels in pregnant solution will decrease the leaching rate and the rate of adsorption on carbon. Oxygen is required in solution to assist with adsorption.

3.3 Ion-Exchange Resin

Ion-exchange resin technology has evolved over the last five years and a number of new gold processing plants have installed resin-in-column (RIC) technology. A major drawback with RIC technology is the selectivity of the resin. In general, resins are known to have a less selective adsorption than carbon. The selectivity with resin-in-column technology may be achieved by selective elution or selective electrowinning. Resins are found to be more efficient than carbon when the concentration of the competing anionic complexes are low, but less efficient when the anions are high.

A strong-base resin is capable of achieving a loading of 2 to 4 times the maximum loading of a weak-base resin. A major drawback with the use of strong-base resins is that elution is not as easily achieved relative to a weak-base resin. In general, a resin has a faster loading and a higher equilibrium capacity than carbon. Loadings of up to 17.8 kg/t of gold on resin have been achieved. Ion exchange resin is known to overcome carbon loading limitations by not adsorbing calcium carbonate, organic matter and clay type minerals. Long stripping times are not required with RIC.

The development of RIC technology is predominantly centered on non-selective adsorption followed by selective elution. A low adsorption efficiency of precious metals and the high cost of eluting agents makes the operating costing higher than zinc precipitation, but because of ambient temperature elution and no thermal regeneration requirements, the capital cost is less than carbon-in-column technology.

3.3.1 Calcium

Resin does not adsorb calcium carbonate. However, a change in the pH may affect the calcium loading.

3.3.2 Base Metals

Base metals such as copper, nickel and zinc (and to a lesser extent iron) can occupy about 95% of the resin capacity, thus limiting the absorption efficiency of precious metals. In some cases, the precious metals may occupy 1 to 2% of the resin. Copper is noted as being a major detriment with most ores and the cost of removing copper may be greater than the cost of recovering gold.

3.3.3 Resin Elution

The resin circuit will require an effective elution agent to strip gold and regenerate the resin. Common elution agents include caustic, ammonium thiocyanide, zinc cyanide or thiourea. About 6 to 10 hours is required for elution from a strong base. Elution may be done at room temperature.

3.3.4 Silica

Silica loading is most prominent using resins and a caustic scrub is required to remove silica from the resin.

3.4 Solvent Extraction

Technological development is predominantly in its infancy for solvent extraction of heap leach solutions. The use of tributyl phosphate increases the apparent basicity of weak base amine solvent which results in an excellent selectivity for gold over other metal cyanides such as copper and iron cyanide. Solvent extraction is commercially used at major base metal smelting

operations to recover gold as a by-product. For solvent extraction of heap leach solutions, efficient clarification of pregnant solution is required.

3.5 Direct Electrowinning

Direct electrowinning is an emerging technology and is technically possible on solutions containing high concentrations of gold. In most cases, a preconcentration of gold is required. For heap leach solutions, which normally contain less than 2 ppm gold, the heap operation will require a solution management program to maintain the pregnant solution at high gold concentrations on a continuous basis.

Direct electrowinning cells must have high surface area electrodes - usually supported fibre flow-through electrodes or packed bed. Design of these electrowinning cells are presently in early stage development. In addition to low mass transfer, cyanide is destroyed in direct electrowinning cells. Selectivity, based on gold-base metal recovery, has not been reported.

4.0 EVALUATION OF OPTIONS FOR GOLD RECOVERY

A process consists of a number of widely varying sequential steps called "circuits". In the case of recovering gold from low grade ores by heap leach processing, the heap leach, gold recovery from solution, solution effluent treatment and smelting of the residue, precipitate or electrodes are considered circuits. Dividing the process into circuits is based on the fundamentals of the process and the operating function of each circuit. The basic operating functions or criteria for the recovery of gold from solution circuit, include the following:

- The treatment of solution containing low concentrations of gold cyanide and other metal cyanides from a heap leach operation.

- The selective extraction of gold from solution and concentrating the gold for precious metal smelting.

- The recycle of cyanide solution to the leaching circuit.

- The production of an effluent which can be subsequently treated by the effluent treatment circuit, thus producing an environmentally acceptable process waste.

An evaluation format was developed to assess the various circuit technologies used to recover gold from heap leach-cyanide solutions. The evaluation criteria was designed to determine the relative maturity of the technology and to define strengths and weaknesses of each technology. Technical feasibility, economic criteria and development status were included as evaluation criteria. Analysis of these factors can provide all concerned - especially project and corporate management - a relative comparison of each technology's commercial probability or utilization risk.

4.1 Evaluation Criteria

The following subsections provide a brief explanation of the criteria utilized in evaluating each of the technologies as well as an indication of their relative contribution (in terms of points) to the development of a final score.

A definite rating method was used to score each of the fifty-four significant project elements. This method employs a weighted-average assessment based on the relative importance of each element to the overall evaluation. The marking scheme for each element was selected as follows:

RISK	CRITERIA
0 to 1	Not Acceptable, High Risk.
2 to 3	Limitations Identified, Solution Unknown
4 to 5	Average/Moderate Risk
6 to 8	Acceptable Industrial Standard
9 to 10	Proven Industrial Standard

The accompanying Table 4.1 defines the weighted averages for each of primary and secondary categories. Details of the tertiary categories and marks assigned to each element is included in Appendix I. Ratings for each category were separately totalled and converted by a multiplier. Each score is relative to the total weighted average mark. The resulting definite rating is a sum of each primary rating and provides a numerical measure of the technology's relative maturity from 0 to 100.

TABLE 4.1

ASSIGNED WEIGHTED AVERAGES FOR ASSESSMENT OF CIRCUIT TECHNOLOGIES

CATEGORY	PRIMARY WEIGHTED AVERAGE (%)	SECONDARY WEIGHTED AVERAGE (%)
1.0 TECHNICAL FEASIBILITY	50	
1.1 Feedstock Characteristics		25
1.2 Operating Efficiency		45
1.3 Raw Materials and Consumables		15
1.4 Mechanical		15
2.0 ECONOMIC CRITERIA	30	
2.1 Labour		40
2.2 Raw Materials and Energy		20
2.3 Capital Cost		30
2.4 Sensitivity		10
3.0 DEVELOPMENT STATUS	20	
3.1 Support of Process Development		30
3.2 Availability of Technical Support		25
3.3 Commercial Applications		15
3.4 Engineering Scale-up Data		15
3.5 Engineering Management		15

4.2 Technical Feasibility

The technical competence of a process system's ability to handle a specific or well defined feedstock is the most important criteria for the selection of circuit technology. A circuit operation must be technically proven before other factors are considered in the selection assessment. In most cases, the technological development has not been fully determined by bench scale and/or pilot scale testing. Optimization studies are generally undertaken on commercial operating plants. Therefore, the level of confidence in appraisal of a technology is generally related to the technologies which are commercially applied to diverse feedstocks. However, the assessment of technical competence contained in this review is based on defined technical performance and includes development status (defined in Section 4.4) as a separate evaluation criteria.

A circuits technology must be capable of maintaining a controllable steady state operation with respect to variations in feedstock characteristics. The various factors used to measure operating efficiency should be known and characterized on specific feedstocks to enable direct steady state control of the process. Both feedstock characteristics and operating efficiency have a major influence on the technical feasibility of a circuit. The control of the technical efficiency of a circuit is also related to the utilization of raw materials, consumables and the mechanical efficiency of the system's equipment. Technical feasibility accounts for 50% of the technology assessment.

4.3 Economic Criteria

The assessment of the economic criteria for a circuit's technology was based on relative production cost data. The economic viability of a circuit can be evaluated if the value of the feedstock and the circuit products are defined. Since various economic factors can not be

identified to establish a return on investment, the assessment includes a relative comparisons of labour costs, consumable costs and capital costs. Included in the assessment of economic criteria is the sensitivity of the circuit's production cost with respect to a variation in feedstock tonnage and composition. The section rating accounts for 30% of the total assessment score.

4.4 Development Status

The development status assessment criteria was used to measure the maturity of the system's technology and the relative probability of commercial use based on established commercial use. The commercial development of the circuit's technology was assessed on the availability of engineering design criteria and piloting and/or commercial optimization studies. Technologies having commercial applications receive a relatively high rating and account for 20% of the total assessment score.

4.5 Summary of Assessment Results

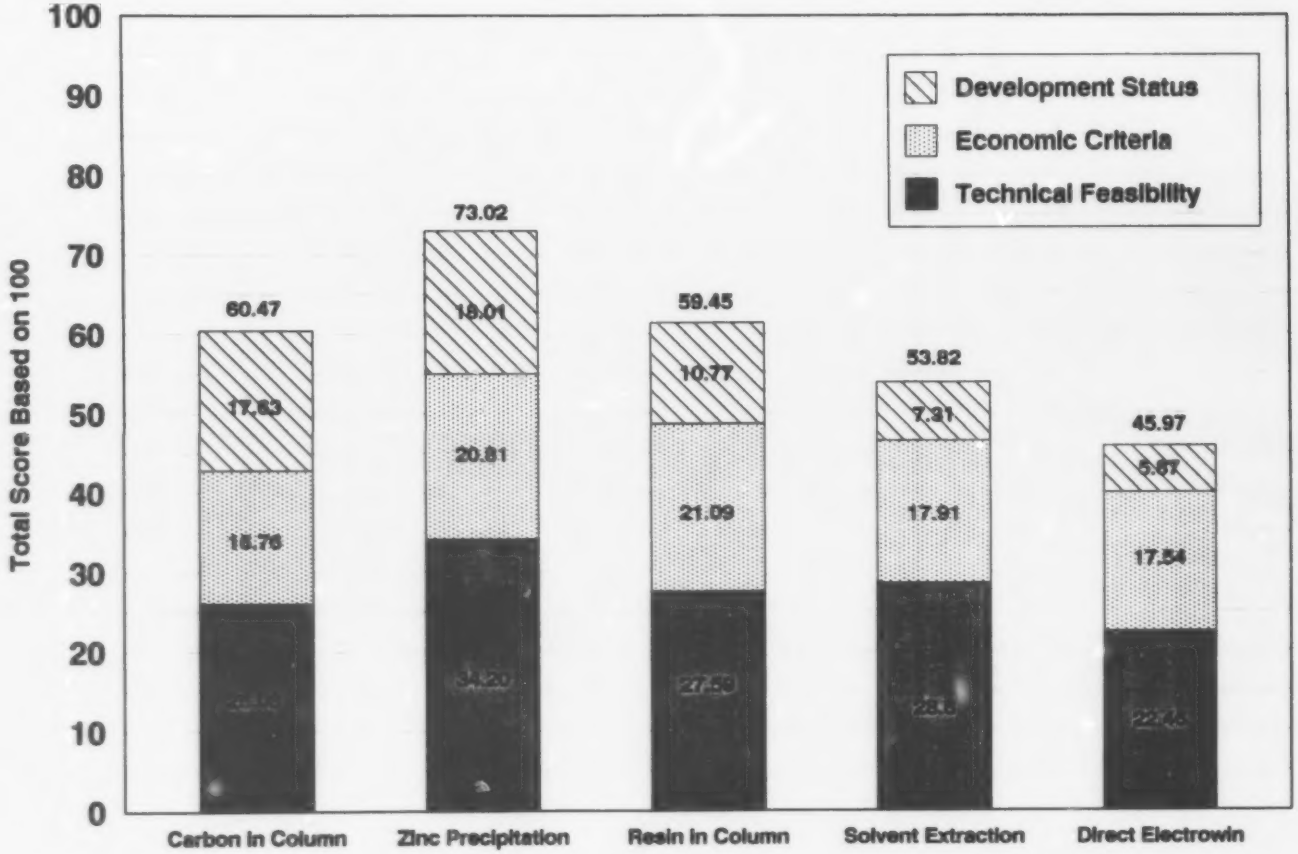
The overall assessment of the circuit's technology was completed with the objective of defining the most appropriate system technologies for a prefeasibility characterization on the Cape Spencer occurrences. Details of the evaluation results are included in Table 4.1 and Appendix I. A summary of the assessment is illustrated by Figure 3.1.

In summary, zinc precipitation and carbon-in-column technologies were judged as being most appropriate for the prefeasibility test program and cost estimation. Resin-in-column may warrant further consideration as more commercial applications are developed. It is anticipated that with increased research and development on resin-in-column technology, this assessment score will exceed other technologies and resin-in-column will emerge as a leading technology. However, for Cape Spencer occurrences, the present day relative comparison of commercial potential for each gold recovery system technology was judged to be as follows:

TECHNOLOGY	RELATIVE COMMERCIAL POTENTIAL
Zinc Precipitation	1.0
Carbon-in-Column	0.83
Resin-in-Column	0.81
Solvent Extraction	0.73
Direct Electrowinning	0.63

ASSESSMENT OF GOLD RECOVERY TECHNOLOGIES

FIGURE 3.1



5.0 ASSESSMENT OF EXTRACTION PLANT CAPACITY AND SIZE

The circulation of aqueous cyanide solution is the major control component of heap leach operations, and the volume of pregnant solution determines the capacity of equipment required in the extraction process plant. As illustrated in Figure 5.1, the solution is transferred from the barren solution pond through the heap to recover precious metals. The pregnant solution is then transferred to the extraction process plant or recycled to the heap via the barren solution pond. With each cycle through the heap (bypassing the extraction plant), the concentration of precious metals increases. However, the addition of water by process water makeup, precipitation and chemical balance, and the removal or storage of water by evaporation, bleed stream and build-up of water inventory, all affect the concentration of precious metals in the pregnant solution and the subsequent size of the extraction plant. Discharge of water from the process can be expensive in terms of both investment for water treatment facilities and for operating costs. Therefore, a zero discharge concept and the management of solution flows must be given special consideration when selecting the extraction plant circuit technology.

5.1 Heap Leach Water Balance

The key aspect for the design of a heap leach processing facility is to determine the overall water balance. A mass balance spreadsheet program was developed to assess the water balance and to determine the extraction plant solution volume for an annual production of 250,000 tonnes per year.

The average water balance includes the process circuit and the natural water circuits. A period over which the average water balance components was based, included the actual leach and rinse time. Water flows related to precipitation and evaporation were based on heap leach area and long-term averages. An overall water balance of the heap leach process (in reference

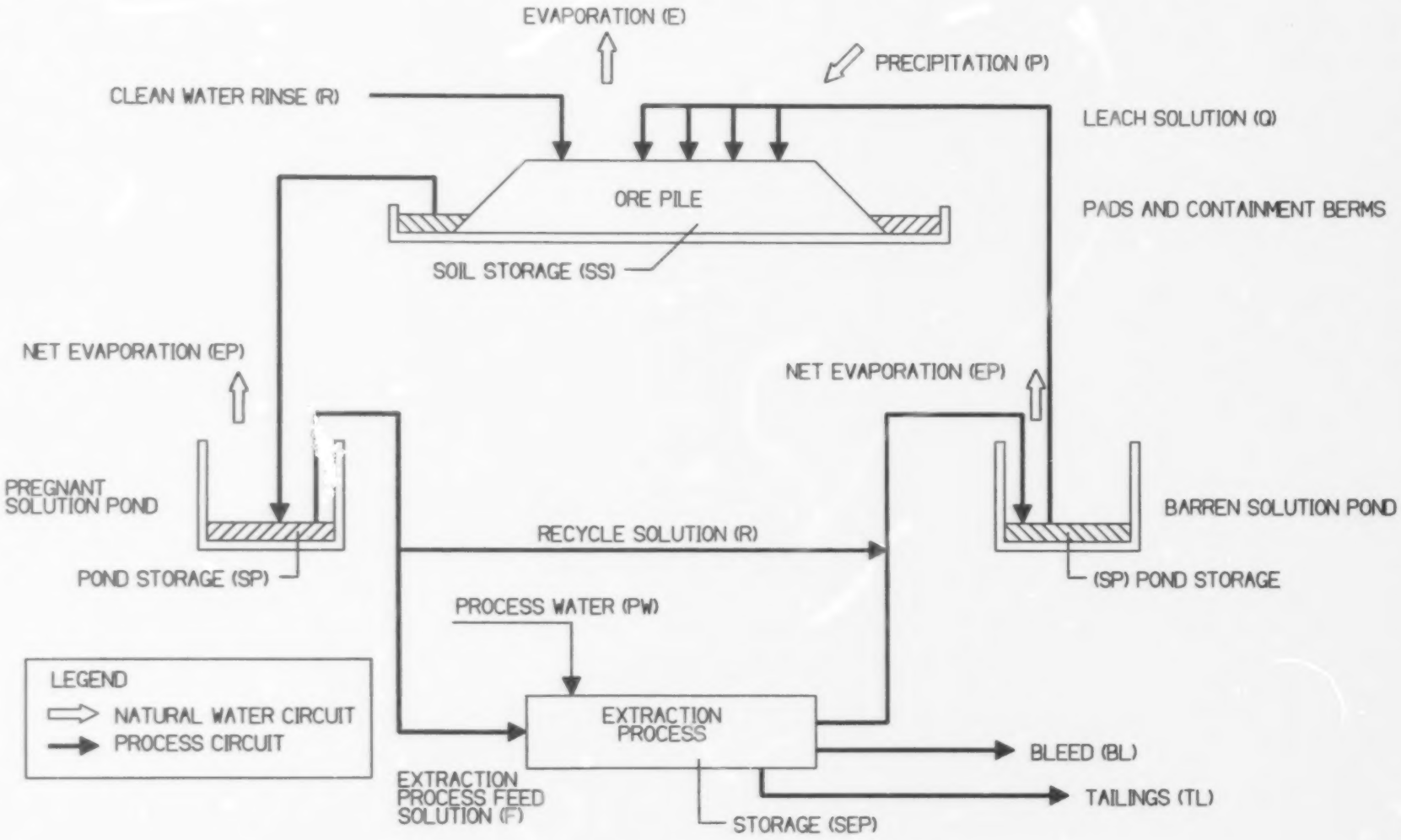


FIGURE 5.1 - HEAP LEACH OPERATION WATER CIRCUITS

to nomenclature shown by Figure 5.1) was calculated in the water balance program by the following equation:

$$PW + P + R = EP_B + SP_B + E + SS + EP + SP_P + BL + TL + SEP$$

The flowrate of recycle (R) and feed (F) to the extraction plant was based on the controlled concentration of precious metal in feed solution. A leach solution application rate (Q) was based on the design application rate for optimum percolation and the area of the heap. Based on the application rate and the recycle rate, solution flowrate to the extraction plant is calculated. The result of the calculations used in the heap leach water balance are included in Appendix II and the design of the water balance program was based on the following criteria:

1) The "total heap leach cycle" or total time required to achieve an optimum extraction of gold was based on the extraction curve illustrated by Figure 5.2. The extraction curve was divided into three zones having a rate of extraction which is dependent on the definition of a zone (area under extraction curve). Zone I has the maximum extraction rate which occurs within the first 2 - 7 days of the leach. Zone II defines the intermediate zone and Zone III defines the tail end of the leach. Rinse time, or the time used to wash the residual gold from the leach pad, was included in Zone III time.

2) Ore is piled on the heap leach pad in cycles and the number of annual cycles is dependant on the leach extraction curve. The heap is divided into an arbitrary number of sections per lift and the number of lifts are selected to determine the total heap area requirements. Bulk density of ore and selection of lift height was used to determine "heap area requirements". Because of the sloping of the ore on the heap, a correction factor of 1.05 is used to determine "corrected heap area".

"Heap loading rate" was related to annual tonnage and total tonnage of ore per cycle.

3) "Solution application flux" was determined by previous testing and based on heap area, the "solution application rate" was calculated.

4) Solution storage pond sizing was based on a total operating volume of solution per day which includes precipitation and solution application rates. Total area of precipitation was based on "active precipitation area factor" which is a multiplication factor for heap leach pad area. Pond sizing also includes "free-board factor" or a fraction of volume used for free-board.

5) To optimize on the concentration of precious metals in pregnant solutions, the heap operation was designed to overlap cycles. Zone III of the initial cycle will overlap with Zone I and Zone II (partially) of the subsequent cycle and is based on the time frame of each zone. Therefore, the solution application rate will double in the overlap time frame since the section area of the heap per cycle has doubled.

6) Rinse water application rate was the same as solution application flux.

7) Amount of water loss to tailings was related to water contained with clarification solids.

8) The amount of water loss due to evaporation was related to the solution application rate and a multiplication factor known as the "heap evaporation loss factor".

9) Pond evaporation rates were based on the pond area and "monthly lake evaporation".

10) Water storage in heap was based on water weight fraction in ore and the maximum weight fraction of water during heap leach operation.

11) The amount of gold in pregnant solution was based on the total gold extracted in each Zone of the extraction curve and the water balance.

12) The amount of recycle solution was based on a set "minimum gold concentration in the pregnant solution", fed to the extraction plant. Water balance calculations and the net demand for recycle determine "extraction plant feed rate".

5.2 Extraction Plant Feedstock Flowrate

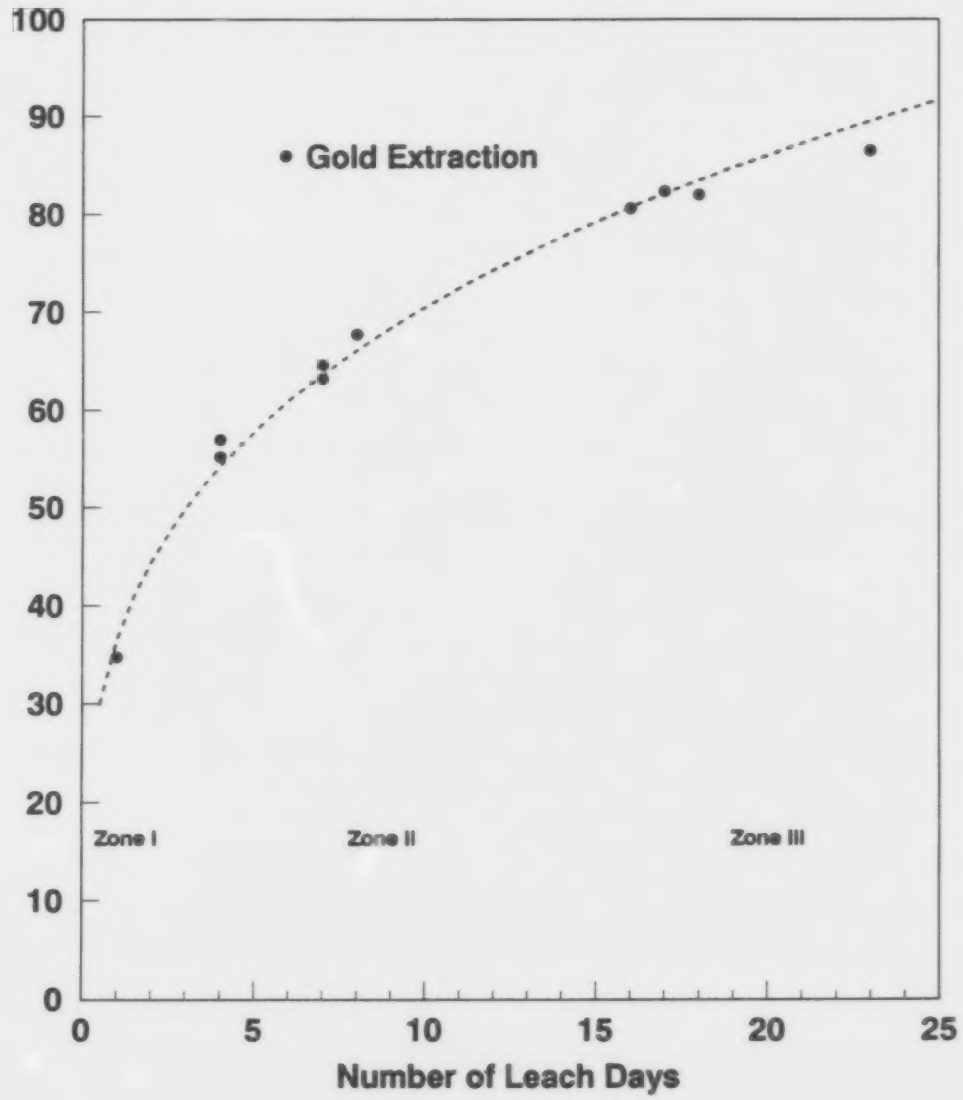
The pregnant solution flowrate was calculated using the water balance program described in the previous sections. Design of the heap leach operation was specified for 250,000 tonnes per year and 340 days of operation. Previous column-leach testing by RPC, on a sample containing 0.60 g/t gold, provided data for the extraction rate and extraction time zones. Solution application rates of 169 lt/hr/m^2 were used in the test program and the resulting extraction curve is shown in Figure 5.2.

A summary of the water balance calculations for two separate cases are reported in Table 5.1 and details of the calculations are included in Appendix II. The difference between Case A and Case B is related to the definition of each extraction zone and the subsequent extraction rates in each zone. The variation in zones reflects the range of pregnant solution flow from 55.4 m^3/hr to 77.3 m^3/hr. Assumptions used in the water balance calculation are as follows:

MISPEC CAPE SPENCER GOLD OCCURANCE

COLUMN LEACH EXTRACTION CURVE ON - 3.8mm CRUSH

FIGURE 5.2



Solution Application Rate:	**169 L/hr/m²**
Head Grade of Gold:	**0.60 g/mt**
Column Solid Loading:	**300 Kg**
Bulk Density of Solids:	**1.157 mt/m³**

TABLE 5.1

SUMMARY OF WATER BALANCE CALCULATIONS

SOLUTION APPLICATION FLUX: 169 lt/hr/m^2

GOLD CONCENTRATION IN PREGNANT SOLUTION: 0.36 g/t

	CASE A	CASE B
TOTAL LEACH CYCLE (DAYS)	25.0	25.0
EXTRACTION TIME (DAYS)		
- ZONE I	1.75	2.5
- ZONE II	15.5	12.0
- ZONE III	8.0	10.5
BARREN SOLUTION APPLICATION RATE (m^3/hr)	543.65	460.46
NUMBER OF LEACH CYCLES	19.22	22.69
EXTRACTION RATE (% PER DAY)		
- ZONE I	31.4	22.4
- ZONE II	1.89	2.08
- ZONE III	0.63	0.57
EXTRACTION PLANT FEED		
NO RECYCLE (m^3/hr)	1085.15	903.45
WITH RECYCLE (m^3/hr)	77.32	55.41

1) Total lift height of leach is 3.67 m.
2) Monthly precipitation averages 88 mm.
3) Monthly lake evaporation averages 100 mm.
4) Solution inventory in ponds remains the same (before and after leach).
5) Active precipitation area factor is 1.15
6) Minimum gold concentration in pregnant solution is 60% of total gold content in the ore.

5.3 Extraction Plant Equipment Sizing

In estimating the cost of an extraction plant using zinc precipitation or carbon-in-column technology, equipment sizing was determined for an average pregnant solution flowrate of 65 m^3/hr containing 0.36 g/t (ppm) gold. Sizing calculations were based on standard order of magnitude methods. Design criteria for zinc precipitation and carbon-in-column are summarized in Table 5.2 and 5.3 respectively. Wherever possible, data from the RPC test program was used as preliminary design criteria. Other design variables were based on published information and generally accepted engineering practice.

The process flow diagrams and conceptual equipment layout drawings for both zinc precipitation and carbon-in-column technology are included in Appendix III. The battery limits of the plant include the pregnant solution intake and the barren solution discharge from the extraction plant. Process flow diagrams and equipment selection only includes the extraction plant circuits. Equipment listings and conceptual process equipment sizes are included in Appendix IV.

TABLE 5.2

DESIGN CRITERIA FOR ZINC PRECIPITATION

CAPACITY:	65 m^3/hr of pregnant solution
SCHEDULE:	Semi-Continuous Operation
CLARIFICATION	
Solid Settling Area in Pregnant Pond:	0.5 to 1.5 m^3/t/day
Solution Filtration Rate:	3.7 $m^3/hr/m^2$
Bulk Density of Clays Filtered:	1.19 t/m^3
Bulk Density of Diatomaceous Earth (DE):	0.317 t/m^3
Precoat Thickness of DE:	1.5 to 3.0 mm
DEAREATION	
Vacuum:	558 to 635 mm Hg
Dissolved Oxygen Concentration:	8 ppm to less than 1 ppm
Vacuum Capacity:	142 to 217 cm^3/sec/lt solution treated
PRECIPITATION	
Zinc Addition:	1 - 2 g/g Au
Lead Addition:	40 g lead nitrate per 450 g Zn
Diatomaceous Earth:	25% to 50% of Zn consumed by weight
Precipitate:	10% to 40% Au

TABLE 5.3

DESIGN CRITERIA FOR CARBON-IN-COLUMN

CAPACITY:	65 m^3/hr of pregnant solution
SCHEDULE:	Continuous Operation
CARBON COLUMN	
Superficial Solution Velocity:	60 m/hr
Height to Diameter Ratio:	6.5:1
Carbon Maximum Loading of Gold:	6,500 g/t
Number of Columns:	1
STRIP VESSEL	
Strip Solution:	0.1% NaOH, 0.1% CN^-
Temperature:	130° C
Flow Rate:	2 to 4 bed volumes per hour
Pressure:	Ambient to 25 psi
Strip Time:	70 hours
Height to Diameter Ratio:	6:1 to 10:1
Superficial Solution Velocity:	7 m/hr
CARBON REGENERATION	
Activation Temperature:	650° C
Residence Time:	10 - 15 minutes
Kiln Loading Basis:	11 kg/hr/m^2
Feed:	50% Carbon Slurry

6.0 EXTRACTION PLANT CONCEPTUAL COST ESTIMATE

This section presents a method for making an economic comparison of zinc precipitation and carbon-in-column circuits. The approach presented was based on conceptual costing procedures for a semi-detailed estimate. All major equipment supply costs were based on budget quotes and certain capital and operating costs were based on standard industrial practice.

Although the total cost obtained using these conceptual cost estimating procedures may not be applicable to a specific facility, the use of the method described should provide a valid relative economic comparison for circuit technology alternatives. The basic assumptions are presented so they can be modified by investor-owned projects.

6.1 Total Capital Requirement

The total capital investment is called the total capital requirement (TCR) for the gold recovery circuit and includes all capital required to supply and install the process facility. A summary of both zinc precipitation and carbon-in-column is included in Table 6.1. The capital cost assumptions were based on the following criteria for an investor contracted supply and installation. Equipment supply costing was not based on a packaged-circuit cost or a turn-key operation.

Direct Costs

Direct cost includes all on-site operation facilities. Details of cost estimation are included in Appendix V for both zinc precipitation and carbon-in-column. Included under direct cost is the equipment purchase, installation, transportation, piping, instrumentation and electrical, site excavation and preparation for process building.

TABLE 6.1

GOLD EXTRACTION PLANT, CAPE SPENCER, NEW BRUNSWICK

ORDER OF MAGNITUDE CAPITAL COST

JANUARY 1990

CE INDEX OF 356.2

BASED ON 65 m^3/hr PREGNANT SOLUTION CONTAINING 0.36 g/t GOLD

	Zinc Precipitation	Carbon In Column
I DIRECT COSTS (DC)		
A. Equipment (Installed)*	$657,400	$556,000
B. Process Building and Site Preparation	175,000	247,500
C. Piping and Installation	21,500	27,000
D. Instrumentation Supply and Installation	45,000	97,500
E. Electrical Supply and Installation	45,000	30,000
TOTAL DIRECT COST	**$943,900**	**$958,000**
II INDIRECT COST (IC)		
A. Material Indirects (8% DC Materials)	$30,500	$23,900
B. Engineering, Design Only	200,000	200,000
C. Construction Management (12% DC Labour)	33,000	30,800
D. Equipment & Field Changes (2% DC)	19,000	19,200
E. Project Contingency (20% DC)	189,000	191,600
TOTAL INDIRECTS	**$471,500**	**$465,500**
III FIXED CAPITAL INVESTMENT	$1,415,400	$1,423,500
IV WORKING CAPITAL (10%)	$141,500	$142,400
V TOTAL CAPITAL INVESTMENT	**$1,556,900**	**$1,565,900**

* Based on trades mix and includes transportation FOB Saint John conceptual.
Estimate accuracy of project cost is +/- 30% based on an estimate.
(Probability of estimate being +/- 10% is 40% to 50%).
Details of cost estimate and omissions reported in text.

The direct cost does not include a lime slaking facility, leaching pads, ponds and equipment necessary for leach operations, mining and crushing equipment, refining, and does not include general off-site capital facilities such as roads, electricity distribution and hookup, buildings, parking lots or laboratories.

Materials Indirects

Materials indirects includes taxes/duties, vendor representatives and surplus materials. Materials indirects is based on 8% of direct cost.

Owner's costs

Was not included in the TCR and will be related to pre-operating cost of construction and owner supervised contract work, licensing, environmental approval, insurance and infrastructure.

Engineering

Engineering costs include salaries, overheads and home office expenses for basic design and project management. Engineering does not include project development, construction follow-up or startup. A total of 2,500 manhours was estimated for (either) circuit.

Construction Management

Installation costs were based on trades mixing included as direct costs. Union labour rates, productivity and overhead for equipment was included in trades manhour cost estimates. Trade supervision, field office expense and payroll burden were included as construction management expenses based on 12% of direct construction labour.

Equipment & Field Changes

Amount of modification to equipment and changes in field installation was estimated at 2% of direct costs because of the low complexity of design.

Escalation

Escalation was not included. If included, the centroid at midpoint of engineering schedule would be approximately 1.5 months from award of contract, and approximately 7 months centroid at 2/3 of construction time.

Project Contingency

Since the cost estimate was based on semi-detailed costing and the design and execution basis depended on limited preliminary work, a project contingency of 20% direct cost was used (based on guidelines for project contingency).

Process Contingency

Since the technology is used commercially, a process contingency was not included.

Working Capital

Supply of consumables and preproduction costs to cover personnel wages is based on 10% of the fixed capital investment.

Project Risk

Cost associated with owner's changes in scope and schedule, which are defined in a risk fund, was not included.

A capital cost estimate was based on a plant sized to handle 65 m^3/hr of pregnant solution containing 0.36 g/t gold. For zinc precipitation and carbon-in-column, the respective total capital requirements are $1,556,900 and $1,565,900 based on January 1990 Canadian

dollars. An estimated accuracy of the project cost was based on the method used for cost estimation and may have a difference of plus or minus 30%. Within the accuracy of the cost estimating methods, the relative total capital requirements are as follows:

Zinc Precipitation:	1.00
Carbon-In-Column:	1.01

6.2 Production Cost

The conceptual production cost was estimated for the first year of operation for zinc precipitation and carbon-in-column. This First Year Production Cost (FYPC), summarized in Table 6.2, includes operating and maintenance cost on an annual basis. The value given can vary depending on many factors, such as plant location, company policies and experienced labour. Guidelines for assigning values to the items are as follows:

Raw Materials

The primary items under this category include reagents based on the conceptual design consumption and supply FOB Montreal. A transportation cost of $0.1123 /t/km was used to determine cost FOB plant site. Reagent and consumable costs are based on supplier cost as follows:

Diatomaceous Earth	$0.75/kg
Zinc	$3.30/kg
Lead Nitrate	$1.6/kg
Caustic (100% basis)	$500/t
Sodium Cyanide	$3.00/kg
Filter Cloth	$10/m^2
Carbon 8x30	$3.05/kg
Sulfuric Acid	$110/t

TABLE 6.2

GOLD EXTRACTION PLANT, CAPE SPENCER, NEW BRUNSWICK

ORDER OF MAGNITUDE ANNUAL PRODUCTION COST

JANUARY 1990

BASED ON 65 m^3/hr PREGNANT SOLUTION CONTAINING 0.36 g/t GOLD

	Zinc Precipitation	Carbon In Column
I DIRECT COST		
A. Raw Materials	$100,000	$41,000
B. Labour	132,500	194,500
C. Direct Supervisory	35,000	35,000
D. Utilities	22,000	38,500
E. Maintenance (2% FCI)	28,300	28,500
F. Supplies (0.5% FCI)	7,000	7,000
G. Laboratory Charges	65,000	65,000
II FIXED CHARGES		
A. Depreciation (10% FCI)	141,500	142,300
B. Taxes (8% Revenue)	140,800	140,800
C. Insurance (0.6% FCI)	8,500	8,500
D. Overhead (30% Labour)	39,750	39,750
III GENERAL EXPENSES		
A. Administration (10% Labour)	13,000	13,000
PRODUCTION COST	**$733,350**	**$753,850**
CONTINGENCY (3%)	22,000	22,600
TOTAL PRODUCTION COST	**$755,350**	**$776,450**
COST PER TONNE ORE	**$3.02**	**$3.11**

Labour

Total labour for operation and routine maintenance jobs was based on approximate crew requirements for 24 hour operation of each technology. Average annual employee wage was based on $30,000/year. A total of 9 employees were estimated for zinc precipitation and 12 employees for carbon-in-column.

Direct Supervisor

One supervisor was assumed adequate for each technology.

Management

Cost for plant management not included.

Utilities

Cost for electrical power was based on $0.05 per kilowatt. Water supply, compressed air, steam and fuel are not included. Equipment selection does not require steam or fuel.

Maintenance

These expenses include materials only and are based on 2% of fixed capital investment (FCI). Cost for contract maintenance was not included.

Supplies

Supplies required to maintain operation, such as safety needs and lubricants, were estimated as 0.5% FCI.

Laboratory Charges

Costs for off-site control assays.

Fixed Charges

All fixed charges are based on percentage of FCI, revenue and labour. Total expected annual revenue for the project was based on $1.73 million. Taxes include Metallic Mineral Taxes and does not include local property taxes. Insurance was based on equipment replacement. Overhead includes employee burden only.

Contingency

A contingency of 3% includes unforeseen events such as price variations.

The product cost estimate was based on a plant operation, 7 days per week, 24 hours per day. Within the relative accuracy of plus or minus 30%, the relative production cost estimate was judged to be as follows:

Zinc Precipitation	1.00
Carbon-In-Column	1.03

The total production cost estimate does not include the following:

- Patents or Royalties
- Rent
- Research and Development
- Distribution and Marketing
- Financing
- Refining, Leaching or Mining Operations

7.0 CONCLUSIONS

The present study does not clearly identify a single technology which is significantly more attractive for the development of the Cape Spencer Occurrences. A limited number of studies have been published on the economic affect of impurities in pregnant solution and a methodology for technology selection. A review of the literature has indicated various technical implications as a result of impurities, however, the economic significance has not been quantified. It is concluded that carbonates and dissolved solids will have an impact on capital expenditures but fouling ions such as copper may only have a direct influence on production costs and gold recovery.

An initial assessment of the technologies has indicated that zinc precipitation is the most viable commercial method. Carbon-in-column technology is favoured for pregnant solutions having high turbidity and high copper concentrations. Recent developments in ion exchange resins and resin-in-column circuits appear to offer a potential for reduced capital costs and perhaps some operating cost improvement. However, additional studies are necessary to achieve an acceptable level of technical risk.

Preliminary cost comparison of conceptual zinc precipitation and carbon-in-column plants for a 250,000 tonnes per year heap leach has yielded a small difference in both capital and production costs. Preliminary testing by RPC does indicate a limited gold recovery with zinc precipitation. Additional testing would be required to confirm and quantify recovery limitations. If in fact neither technology poses a technical risk and no real cost distinction between the two is evident. A technology selection decision can thus be based on project and operating plant personnel experience.

In addition to technology selection for gold recovery from pregnant solutions, solution management was defined as an integral part of the optimum extraction plant sizing. Initial

column leach tests suggest a high rate of gold extraction but the extraction rate for lower solution application rates may not be similar. The improvement in process economics for lower solution application rates was not quantified.

8.0 RECOMMENDATIONS

Based on a review of published technical literature and an assessment of test data, it is recommended that additional continuous column leach testing be conducted to quantify solution fouling and gold recoveries using zinc precipitation and carbon-in-column technologies. The major recommendations arising from the present assessment can be summarized as follows:

- The present study gives a base line cost for gold extraction plants but does not include mining, leaching, smelting or environmental costs. Prior to conducting pilot plant testing, the cut off grade should be identified, so that technological development is conducted on ores of economic significance.

- The affect of solution application rates on gold extraction and the economic impact on capital and production costs should be included in additional studies.

- An assessment of resin-in-column technology should be conducted to establish the technical performance of this technology and to identify the potential savings over zinc precipitation and/or carbon-in-column.

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APPENDIX I

Assessment of Circuit Technologies Based on Evaluation Criteria

THIBAULT & ASSOCIATES INC. FILE: UNIEVA SPREADSHEET

ASSESSMENT OF CIRCUIT TECHNOLOGIES BASED ON WEIGHT AVERAGED EVALUATION CRITERIA

REVISION: B DATE: 90-03-20	WTD. AVG	CARBON ABSORP	Zn. PRECIP	I/X RESINS	S/X EXTRACT	ELECTRO WINN
1.0 TECHNICAL FEASIBILTY (50%)	50	26.08	34.20	27.59	28.60	22.46
1.1 FEEDSTOCK CHARACTERISTICS	25	14.12	13.88	13.63	16.25	7.50
1.1.1 SOLUTION FOULING	20	7	3	8	5	2
1.1.2 IMPURTIES AND SELECTIVITY OF OPERATION	25	4	3	4	7	4
1.1.3 THROUGHPUT CAPACITY BASED ON CONVERSION KINETICS	10	5	8	5	7	3
1.1.4 SENSITIVITY TO CONCENTRATION OF METALS	40	6	8	5	7	3
1.1.5 DEPENDENCY ON UP-STREAM PROCESSING	5	7	4	7	5	2
1.2 OPERATING EFFICIENCY	45	23.63	31.05	22.95	22.95	19.35
1.2.1 METAL RECOVERY AND GRADE	25	7	7	5	5	2
1.2.2 IMPURITY REJECTION	20	4	3	4	7	2
1.2.3 COMPLEXITY OF OPERATING VARIABLES	15	6	8	5	3	5
1.2.4 CONTROL OF STEADY STATE MASS BALANCE	10	4	8	6	5	4
1.2.5 CONTROL OF REACTION KINETICS AND THERMODYNAMICS	10	6	8	3	3	8
1.2.6 INVENTORY OR HOLD-UP CAPACITY	10	3	9	7	7	9
1.2.7 DOWNSTREAM PROCESSING REQUIREMENTS	5	4	8	6	7	5
1.2.8 NEED FOR RECYCLE STREAMS	5	6	9	8	3	6
1.3 RAW MATERIALS AND CONSUMABLES	15	7.43	10.88	8.25	9.83	9.08
1.3.1 ENERGY DEMAND	35	4	8	6	6	3
1.3.2 REAGENT CONSUMPTION AND PREPARATION	15	8	4	5	6	9
1.3.3 UTILITY REQUIREMENTS	25	3	7	6	8	8
1.3.4 AVAILABILITY, SOURCE AND ALTERNATIVES	15	8	8	3	5	9
1.3.5 DEMAND FOR REPLACEMENT PARTS	10	4	9	7	8	3
1.4 MECHANICAL	15	6.98	12.60	10.35	8.18	9.00
1.4.1 SPECIALIZATION OF EQUIPMENT	10	6	8	7	7	3
1.4.2 OPERATOR FRIENDLY	25	7	9	7	6	9
1.4.3 FLUID TRANSFER CAPABILITIES	15	3	8	6	5	8
1.4.4 MAINTENANCE REQUIREMENTS	35	4	8	8	5	3
1.4.5 AUTOMATION AND INSTRUMENTATION	15	3	9	5	5	8
2.0 ECONOMIC CRITERIA	30	16.76	20.81	21.09	17.91	17.54
2.1 LABOUR	40	20.80	32.80	25.20	24.00	26.00
2.1.1 OPERATOR REQUIREMENTS	40	4	8	6	6	8
2.1.2 MAINTENANCE REQUIREMENTS	30	4	8	7	6	5
2.1.3 CONTRACT EMPLOYEES	20	8	9	5	5	5
2.1.4 ADMINSTRATIVE STAFF	10	8	8	8	8	8

2.2 RAW MATERIALS AND ENERGY	20	12.60	10.60	13.60	14.40	11.60
2.2.1 COST TO PURCHASE	40	5	3	6	6	2
2.2.2 TRANSPORTATION COST	30	8	8	8	8	8
2.2.3 STORAGE COST	20	7	5	7	8	9
2.2.4 UTILITY COST	10	5	7	6	8	8
2.3 CAPITAL COST	30	15.45	18.45	24.00	15.30	17.85
2.3.1 COST TO PURCHASE OR FABRICATE	30	6	5	8	4	3
2.3.2 INSTALLATION COST	25	6	5	8	7	7
2.3.3 TRANSPORTATION COST	15	3	8	8	5	6
2.3.4 PLANT AREA REQUIREMENTS	20	3	7	8	3	8
2.3.5 DUTY AND TARIFFS	10	8	8	8	8	8
2.4 SENSITIVITY	10	7.00	7.50	7.50	6.00	3.00
2.4.1 VARATION OF FEED THROUGHPUT TONNAGE	50	8	7	8	4	3
2.4.2 VARATION OF FEED CONCENTRATIONS	50	6	8	7	8	3
3.0 DEVELOPMENT STATUS (20%)	20	17.64	18.02	10.77	7.31	5.97
3.1 SUPPORT OF PROCESS DEVELOPMENT	30	24.90	27.00	20.10	10.20	8.70
3.1.1 DOCUMENTATION OF TECHNICAL DEVELOPMENT	50	8	9	7	4	3
3.1.2 BENCH AND PILOT PLANT STUDIES	20	8	9	7	4	4
3.1.3 COMMERCIAL OPTIMIZATION STUDIES	30	9	9	6	2	2
3.2 AVAILABILITY OF TECHNICAL SUPPORT	25	22.50	22.50	12.75	9.00	6.75
3.2.1 PATENT(S) RIGHTS	30	9	9	3	5	2
3.2.2 CONSULTATION BY DEVELOPERS	70	9	9	6	3	3
3.3 COMMERCIAL APPLICATIONS	15	13.50	12.90	6.60	3.30	3.30
3.3.1 NUMBER OF COMMERCIAL APPLICATIONS	40	9	8	4	2	1
3.3.2 SCALE OF COMMERCIAL OPERATIONS	20	9	9	4	1	3
3.3.3 SIMILARITY IN FEEDSTOCK CHARACTERISTICS	20	9	9	3	3	3
3.3.4 SIMILARITY IN PROCESS APPLICATION	20	9	9	7	3	3
3.4 ENGINEERING SCALE-UP DATA	15	13.05	13.05	5.40	6.90	5.10
3.4.1 DESIGN VARIABLES	40	8	8	3	5	5
3.4.2 PHYSICAL AND CHEMICAL DATA	30	9	9	3	5	3
3.4.3 COMPUTER SIMULATION	10	8	8	1	1	1
3.4.4 RISK FACTORS	20	10	10	7	5	2
3.5 ENGINEERING MANAGEMENT	15	14.25	14.63	9.00	7.13	6.00
3.5.1 EXPERIENCE IN CONSTRUCTION PROCUREMENT	50	10	10	5	4	2
3.5.2 EXTENT OF ENGINEERING REQUIRED	25	10	10	5	4	4
3.5.3 CONSTRUCTION CONTRACTOR REQUIREMENTS	25	8	9	9	7	8
TOTAL RATING		60.47	73.02	59.45	53.82	45.97

APPENDIX II

Steady State Water Balance Calculations

THIBAULT & ASSOCIATES INC.

STEADY STATE WATER BALANCE FOR HEAP LEACH OPERATIONS

GENERAL	CASE A	CASE B
ANNUAL HEAP TONNAGE:	250000.00	250000.00
DAYS OF OPERATION (DAYS):	340.00	340.00
AVAILABILITY:	0.96	0.96
TOTAL GOLD RECOVERY:	0.87	0.87
ORE GRADE (gm/mt):	0.60	0.60
ANNUAL PRODUCTION RATE (troy oz):	4027.78	4027.78
GOLD VALUE ($CDN/OZ):	$430.00	$430.00
TOTAL REVENUE:	1.73E+06	1.73E+06
HEAP OPERATION		
TOTAL LEACH CYCLE (DAYS):	25.00	25.00
EXTRACTION TIME ZONE I (days):	1.75	2.50
EXTRACTION TIME ZONE II (days):	15.50	12.00
EXTRACTION TIME ZONE III (days):	8.00	10.50
RINSE TIME (DAYS):	0.50	0.50
COMMUNITION SIZE (mm):	3.80	3.80
BULK DENSITY OF HEAP (mt/m3)	1.16	1.16
LIFT HEIGHT (m):	3.67	3.67
NUMBER OF LIFTS:	2.00	2.00
TOTAL NUMBER OF SECTIONS PER LIFT:	4.00	4.00
HEAP AREA REQUIREMENTS (m2):	29438.16	29438.16
CORRECTED HEAP AREA (m2):	30910.06	30910.06
SECTION AREA (m2):	7727.52	7727.52
NUMBER OF LEACH CYCLES (BASED ON ZONE I&II):	19.22	22.69
HEAP SECTION REQUIREMENTS PER CYCLE:	0.42	0.35
TONNAGE PER CYCLE (mt):	13009.05	11018.24
HEAP LOADING RATE (mt/day):	754.15	759.88
SOLUTION APPLICATION FLUX (lt/hr/m2):	169.00	169.00
CYCLE SOLUTION APPLICATION RATE (m3/hr):	543.65	460.46
POND SIZEING - BASED ON MAXIMUM OPERATING VOLUME		
MAXIMUM DAILY OPERATING VOLUME (m3):	26095.37	22101.92
MONTHLY PRECIPITATION (mm):	88.00	88.00
ACTIVE PRECIPITATION AREA FACTOR:	1.66	1.66
PRECIPITATION VOLUME (m3):	4515.34	4515.34
ACTIVE POND VOLUME (m3):	30610.71	26617.26
ACTIVE POND DEPTH (m):	2.50	2.50
POND AREA (m2):	12244.28	10646.90
POND FREE BOARD FACTOR:	0.20	0.20
FREE BOARD DEPTH (m):	0.63	0.63
TOTAL POND VOLUME (m3):	38263.39	33271.57
MINIUM SOLUTION STORAGE (m3):	7488.60	6511.65

TOTAL WATER BALANCE - HEAP LEACH CIRCUIT		
TIME PERIOD (days):	25.00	25.00
OVERLAP TIME PER CYCLE (days):	8.50	11.00
NON-OVERLAP TIME (days):	8.75	3.50
RINSE WATER (m3):	6523.84	5525.48
TOTAL PRECIPITATION (m3):	3641.40	3641.40
SOLIDS-PREG'T SOL'N OVERLAP ZONE (fraction):	0.0150	0.0150
SOLID-PREGNANT SOL'N ZONE II (fraction):	0.0025	0.0025
FRACTION OF MOISTURE - CLARIFICATION TAILS:	0.20	0.20
SPECIFIC GRAVITY OF SOLIDS:	2.50	2.50
CLARIFICATION SOLIDS TONNAGE (mt):	3643.22	3776.78
WATER IN TAILINGS (m3):	910.81	944.19
HEAP EVAPORATION LOSS FACTOR:	0.05	0.05
HEAP EVAPORATION (m3):	22418.91	24392.69
MONTHLY LAKE EVAPORATION (mm):	100.00	100.00
PONDS EVAPORATION VOLUME (m3):	1974.88	1717.24
EXTR PLANT INITIAL INVENTORY (m3):	1000.00	1000.00
EXTR PLANT FINAL INVENTORY (m3):	1000.00	1000.00
BARREN POND INITIAL INVENTORY (m3):	7488.60	6511.65
BARREN POND FINAL INVENTORY (m3):	7488.60	6511.65
PREGNANT POND INITIAL INVENTORY (m3):	7488.60	6511.65
PREGNANT POND FINAL INVENTORY (m3):	7488.60	6511.65
WATER IN ORE TO HEAP (fraction):	0.08	0.08
WATER STORAGE IN HEAP (fraction):	0.12	0.12
ORE WATER STORAGE (m3):	735.29	735.29
EXTR PLANT BLEED STREAM (m3):	0.00	0.00
WATER BALANCE / PROCESS WATER DEMAND (m3):	15874.65	18622.53

TOTAL WATER IN SYSTEM - START UP REQUIREMENTS		
ORE WATER STORAGE (m3):	520.36	440.73
POND STARTUP VOLUME (m3):	14977.21	13023.29
EXTR PLANT STARTUP VOLUME (m3):	1000.00	1000.00
TOTAL START UP VOLUME (m3):	16497.57	14464.02

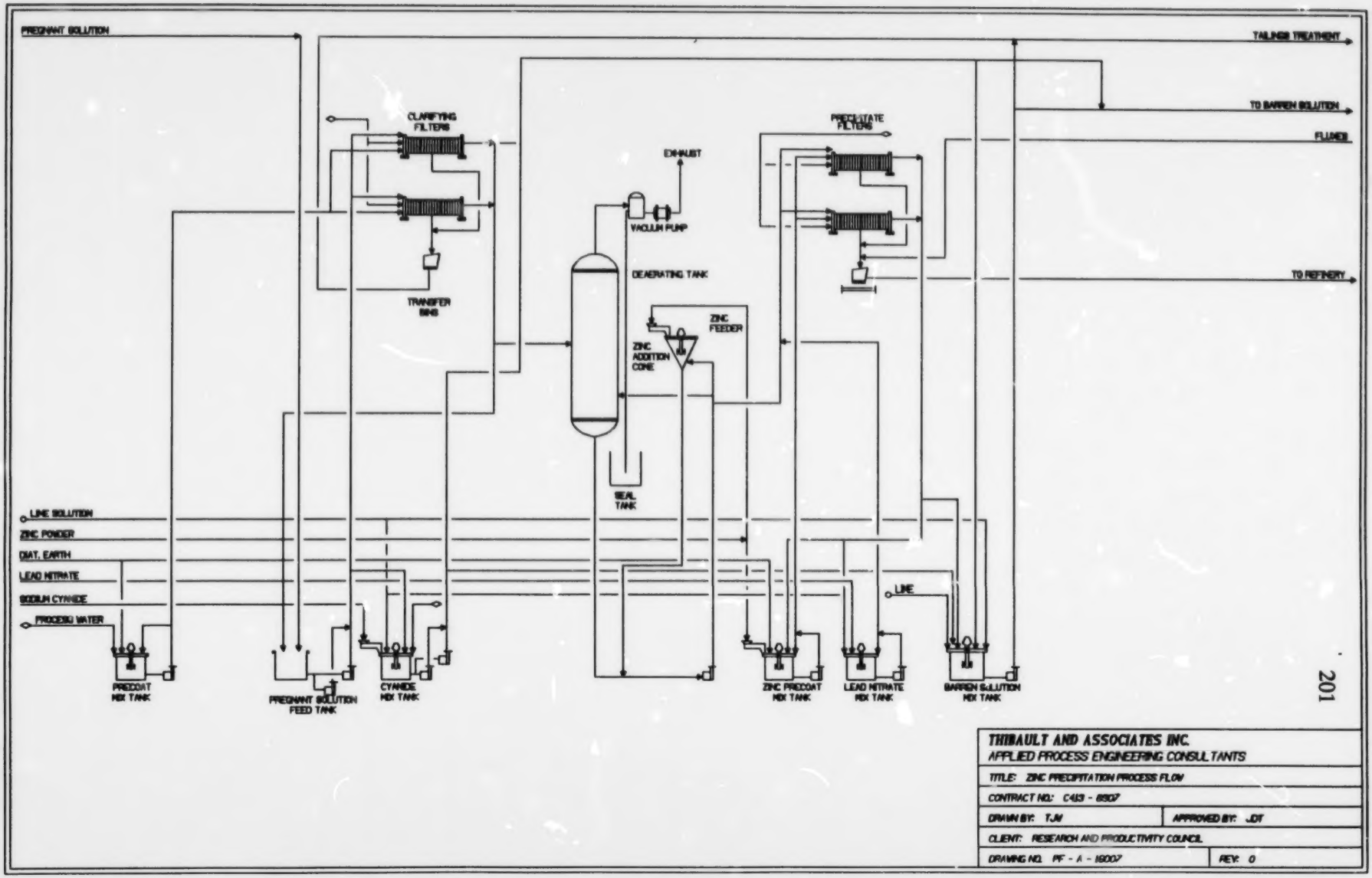
GOLD EXTRACTION RATES AND CONCENTRATIONS		
EXTRACTION RATE - ZONE I (%/day):	31.40	22.40
EXTRACTION RATE - ZONE II (%/day):	1.89	2.08
EXTRACTION RATE - ZONE III (%/day):	0.63	0.57
GOLD EXTRACTED IN OVERLAP PERIOD (gm):	5678.26	5266.61
GOLD EXTRACTED IN NONOVERLAP PERIOD (gm):	1290.82	481.28
MAX. Au CONCENTRATION IN OVERLAP (gm/mt):	0.03	0.02
MAX. Au CONCENTRATION IN NONOVERLAP (gm/mt):	0.01	0.01

MAXIMUM EXTRACTION PLANT WATER THROUGHPUT BASED ON OVERLAP PERIOD

MINIMUM Au CONC. IN PREG'T SOLUTION (gm/mt):	0.36	0.36
TOTAL SOLUTION TO HEAP (m3):	221810.61	243121.08
PRECIPITATION ON HEAP AND POND (m3):	1041.27	1297.65
RINSE WATER (m3):	6523.84	5525.48
POND AND HEAP EVAPORATION (m3):	7958.16	11110.58
HEAP AND POND ACCUMULATION (m3):	250.00	323.53
TOTAL SOLUTION FROM HEAP (m3):	221167.57	238510.10
Au CONCENTRATION IN PREG'T SOLUTION (gm/mt):	0.03	0.02
EXTRACTION PLANT FEED (no recycle)(m3/hr):	1084.15	903.45
CALCULATED RECYCLE VOLUME (m3):	205394.64	223880.64
EXTRACTION PLANT FEED (with recycle)(m3/hr):	77.32	55.41

APPENDIX III

Conceptual Process Flow Diagrams and Equipment Layouts



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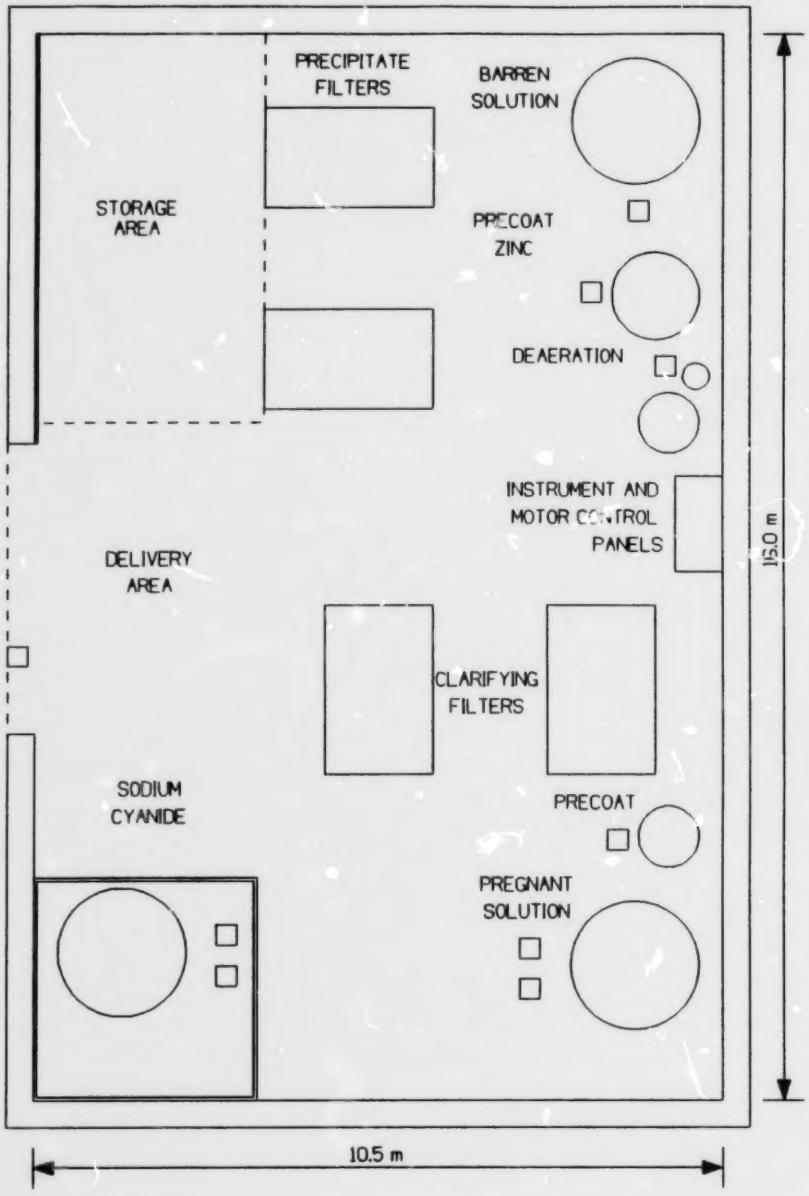

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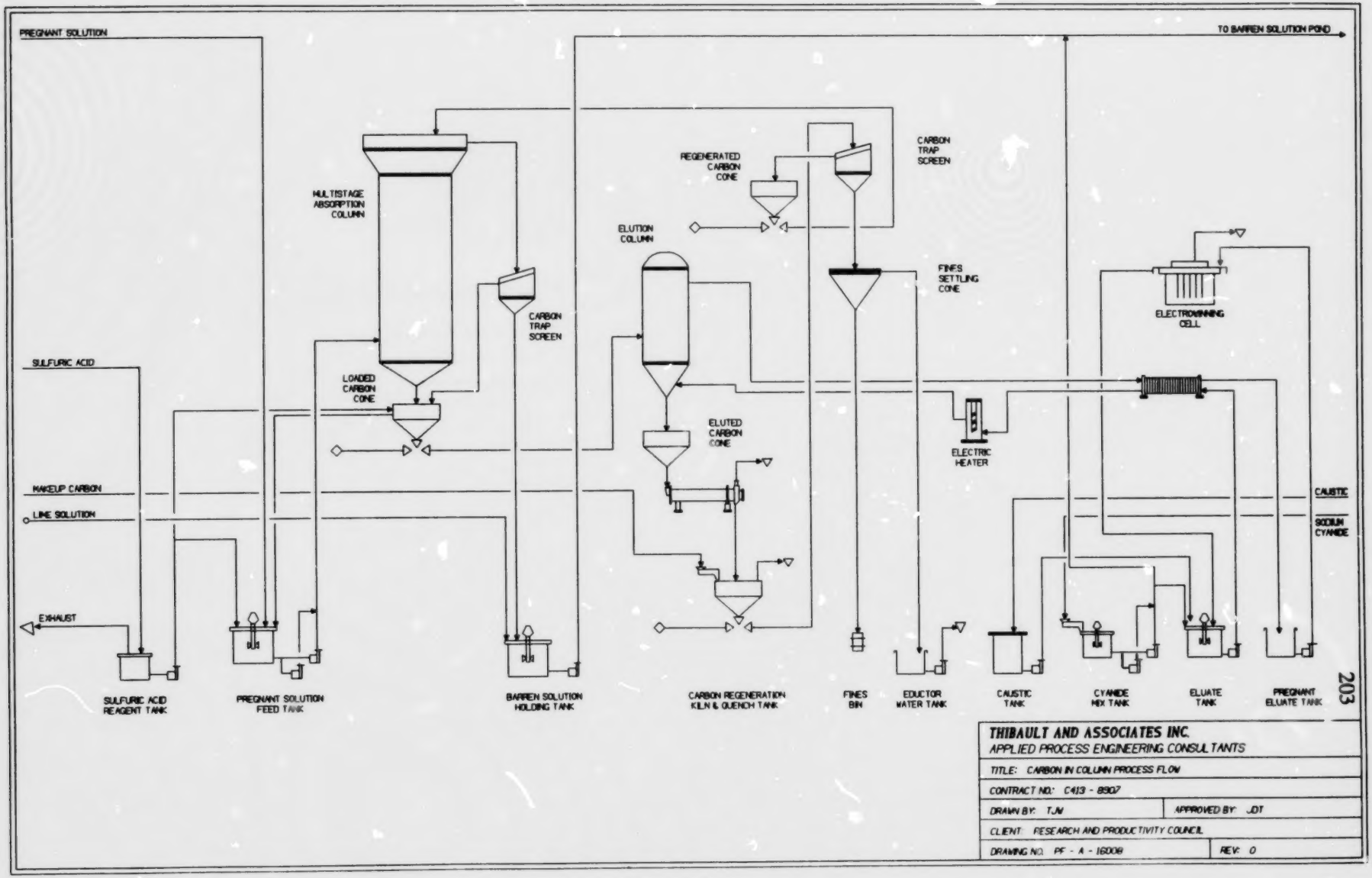
PREGNANT SOLUTION
TO BARREN SOLUTION POND
MULTISTAGE ABSORPTION COLUMN
REGENERATED CARBON CONE
CARBON TRAP SCREEN
ELUTION COLUMN
FINES SETTLING CONE
ELECTROWINNING CELL
CARBON TRAP SCREEN
LOADED CARBON CONE
ELUTED CARBON CONE
ELECTRIC HEATER
SULFURIC ACID
MAKEUP CARBON
LIME SOLUTION
CAUSTIC
SODIUM CYANIDE
EXHAUST
SULFURIC ACID REAGENT TANK
PREGNANT SOLUTION FEED TANK
BARREN SOLUTION HOLDING TANK
CARBON REGENERATION KILN & QUENCH TANK
FINES BIN
EDUCTOR WATER TANK
CAUSTIC TANK
CYANIDE MIX TANK
ELUATE TANK
PREGNANT ELUATE TANK
203
THIBAULT AND ASSOCIATES INC.
APPLIED PROCESS ENGINEERING CONSULTANTS
TITLE: CARBON IN COLUMN PROCESS FLOW
CONTRACT NO: C413 - 8907
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CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL
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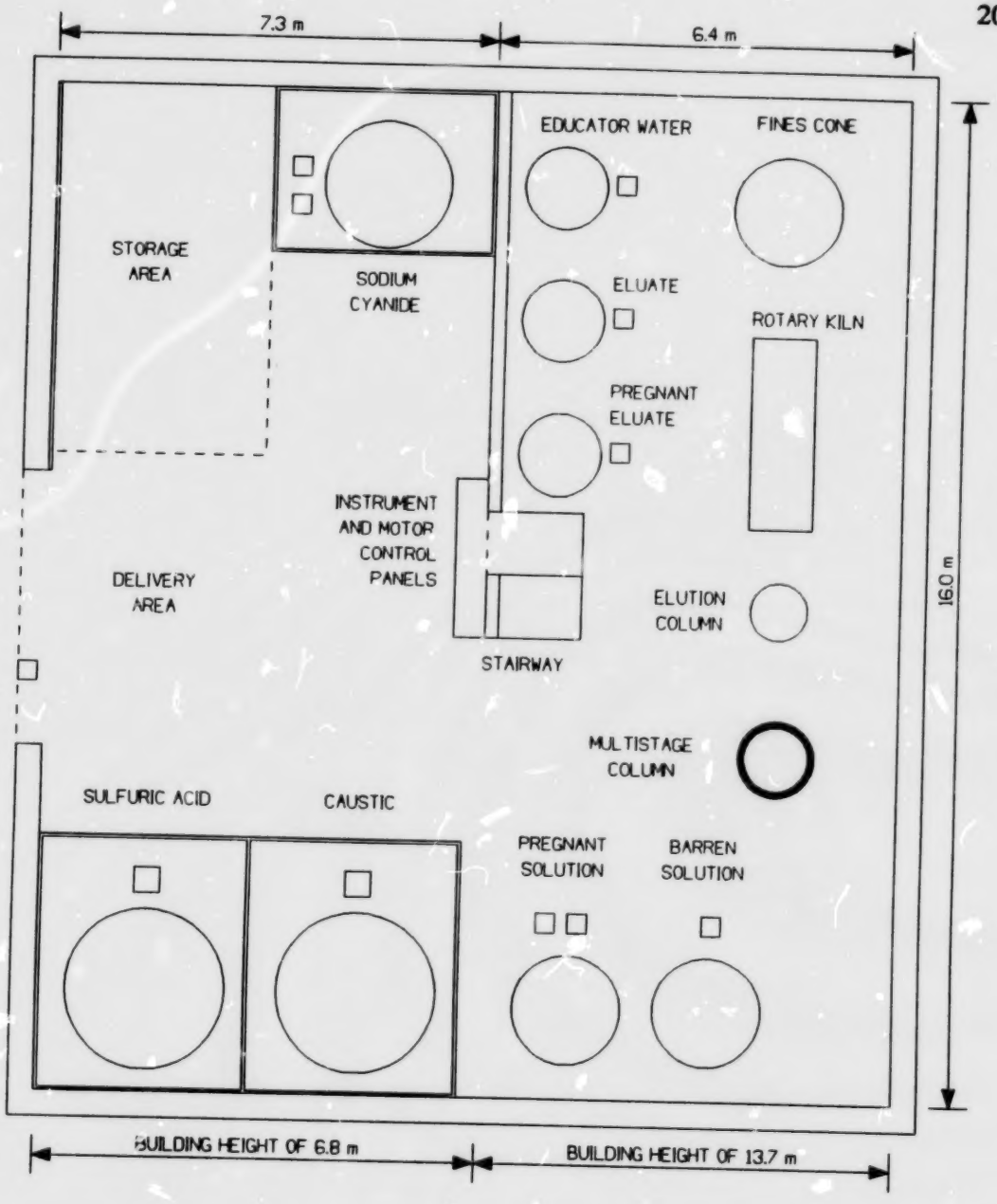

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APPENDIX IV

Mechanical Equipment Index

Thibault & Associates Inc.
Applied Process Engineering Consultants

MECHANICAL EQUIPMENT INDEX

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: ZINC PRECIPITATION

EQUIP.NO.	SIZE	EQUIPMENT DESCRIPTION	SUPPLIER	MAT'L
FP-01	43" PLATE	PREGNANT SOLUTION CLARIFICATION PRESSURE FILTER	SHRIVER	POLYP.
FP-02	"	" "	"	"
FP-03	43" PLATE	BARREN SOLUTION/ ZINC PRECIPITATE PRESSURE FILTER	"	"
FP-04	"	" "	"	"
TS-01	10 m^3	PREGNANT SOLUTION FEED TANK	M.F.F.	POLYP.
TS-02	2 m^3	PRECOAT MIX TANK	"	"
TS-03	10 m^3	CYANIDE MIX TANK (20% solution)	"	"
TS-04	1 m^3	DEAERATION VACUUM SEAL WATER TANK	"	"
TS-05	3 m^3	ZINC PRECOAT MIX TANK	"	"
TS-06	20 Lt.	LEAD NITRATE MIX TANK (5% Solution)	"	"
TS-07	10 m^3	BARREN SOLUTION AND REAGENT ADDITION TK.	"	"
MG-01	1.0 hp	PRECOAT MIX TANK AGITATOR	PROCHEM	C.S.
MG-02	3.0 hp	CYANIDE MIXING TANK AGITATOR	"	"
MG-03	1.0 hp	ZINC ADDITION CONE / MIXER AGITATOR	"	"
MG-04	0.25 hp	LEAD NITRATE MIX TANK AGITATOR	"	"
MG-05	1.0 hp	ZINC PRECOAT MIX TANK AGITATOR	"	"
MG-06	3.0 hp	BARREN SOLUTION TK. AGITATOR	"	"
PB-01	85 m^3/hr	PREGNANT SOLUTION - FILTER FEED PUMP	ALLIS CHALMERS	C.S.
PB-02	"	" " (SPARE)	"	"
PB-03	85 m^3/hr	DEAERATION TANK U/F PUMP	"	"
PB-04	0.51 m^3/hr	ZINC PRECIPITATE PRECOAT FEED PUMP	"	"
PB-05	85 m^3/hr	BARREN SOLUTION TRANSFER PUMP	"	"

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MECHANICAL EQUIPMENT INDEX

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: ZINC PRECIPITATION

EQUIP.NO.	SIZE	EQUIPMENT DESCRIPTION	SUPPLIER	MAT'L
PB-06	0.56m³/hr	CYANIDE REAGENT PUMP	ALLIS CHALMERS	C.S.
PB-07	"	" " (SPARE)	"	"
PB-08	0.51m³/hr	PRECOAT MIX TANK REAGENT PUMP	"	"
TB-01	1.5m³	CLARIFICATION SLUDGE TRANSFER BIN	LOCAL SHOP	C.S.
TB-02	1.5m³	ZINC PRECIPITATE TRANSFER BIN	"	C.S.
SP-01	0.60mϕ	DEAERATION TOWER	"	C.S.
GS-01	2.3m	ZINC ADDITION MIXING CONE (60° Slope)	"	C.S.
TK-01	0.5m³	VACUUM RECEIVER TANK	"	POLYP.
PF-01	6hp	DEAERATION VACUUM PUMP	SIHI	C.S.
CA-01	0.25hp	CYANIDE REAGENT FEEDER	WALLACE & TIERNAN	C.S.
CA-02	0.25hp	ZINC ADDITION CONE FEEDER	"	"
CA-03	0.25hp	ZINC PRECOAT MIX TK. FEEDER	"	"

DATE: MARCH, 1990	BY: D. THIBAULT	PROJECT NO: 8907	REV: 0	PAGE: 2 OF: 2

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MECHANICAL EQUIPMENT INDEX

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: CARBON IN COLUMN

EQUIP. NO.	SIZE	EQUIPMENT DESCRIPTION	SUPPLIER	MAT'L
SQ-01	1.2mϕ x 7.7m	MULTISTAGE GOLD/CARBON ADSORPTION COLUMN	LOCAL SHOP	C.S.
SQ-02	0.76mϕ x 5.7m	SINGLE STAGE CARBON ELUTION COLUMN	"	"
SS-01	$1m^2$	LOADED CARBON TRAP SCREEN	"	"
SS-02	$1m^2$	REGENERATED CARBON TRAP SCREEN	"	"
GS-01	$1m^3$	LOADED CARBON HOLDING CONE	"	"
GS-02	$1m^3$	REGENERATED CARBON HOLDING CONE	"	"
GS-03	$1m^3$	CARBON FINES SETTING CONE	"	"
GS-04	$1m^3$	ELUTED CARBON HOLDING CONE	"	"
GS-05	$1m^3$	REGENERATED CARBON QUENCH CONE	"	"
TS-01	$18m^3$	SULFURIC ACID REAGENT TANK	MFM	C.S.
TS-02	$18\ m^3$	CAUSTIC REAGENT TANK	"	"
TS-03	$10\ m^3$	BARREN SOLUTION HOLDING TANK	M.F.F.	POLYP.
TS-04	$10\ m^3$	EDUCTOR WATER TANK	"	"
TS-05	$10m^3$	PREGNANT SOLUTION FEED TANK	"	"
TS-06	$10m^3$	CYANIDE MIX TANK	"	"
TS-07	$3\ m^3$	ELUATE HOLDING TANK	"	"
TS-08	$3m^3$	PRENANT ELUATE HOLDING TANK	"	"
PB-01	$65m^3/hr$	PREGNANT SOLUTION FEED PUMP	ALLIS CHALMERS	C.S.
PB-02	$6Em^3/hr$	" " (SPARE)	"	"
PB-03	$65m^3/hr$	BARREN SOLUTION PUMP	"	"
PB-04	$3.m^3/hr$	EDUCTOR WATER PUMP	"	"
PB-05	$3.2m^3/hr$	ELUATE FEED PUMP	"	"

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MECHANICAL EQUIPMENT INDEX

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: CARBON IN COLUMN

EQUIP.NO.	SIZE	EQUIPMENT DESCRIPTION	SUPPLIER	MAT'L
PB-06	3.2 m^3/hr	PREGNANT ELUATE FEED PUMP	ALLIS CHALMERS	C.S.
PM-01	0.51 m^3/hr	SULFURIC ACID REAGENT PUMP	"	S.S.
PM-02	0.51 m^3/hr	CAUSTIC REAGENT PUMP	"	C.S.
PM-03/04	0.51 m^3/hr	CYANIDE REAGENT PUMP (SPARE)	"	C.S
EQ-01	0.12m ϕ x 1.2m	CARBON REGENERATION ROTARY KILN.	CE RAYMOND	-
EH-01	6 Kw	ELECTRIC IMMERSION HEATER	CHROMALOX	COPPER
EE-01	1.6 m^3	ELECTROWINNING CELL		
EE-02	"	"		
MG-01	3.0 hp	PREGNANT SOLUTION TK. AGITATOR	PROCHEM	C.S.
MG-02	3.0 hp	BARREN SOLUTION TK. AGITATOR	"	"
MG-03	3.0 hp	CYANIDE MIX TK. AGITATOR	"	"
MG-04	1.0 hp	ELUTE TK. AGITATOR	"	"
HQ-01		ELUTE SOLUTION HEAT EXCHANGER	ALFA-LAVAL	C.S.
CA-01	0.25 hp	MAKEUP CARBON FEEDER	PENNWALT	C.S
CA-02	0.25 hp	CYANIDE REAGENT FEEDER	"	"

APPENDIX V

Conceptual Cost Estimates

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: ZINC PRECIPITATION

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
05-001	PRESSURE FILTER, TOTAL FILTRATION	4	25 500	65 000	362 000
	AREA OF 80m² USING 34 CHAMBERS				
	PLATE PRESSURE FILTER SHRIVER				
	MODEL #42 C/W POLYPROPYLENE				
	PLATES & FRAME WITH C.S. STRUCTURAL				
	STEEL SUPPORT FRAME, HYDRO-				
	KLOSER HYDRAULIC RAM (MANUAL				
	OVERRIDE) AND LINK LATCH SHIFTING				
	DEVICE. [FP-01/-02/-03/-04]				
05-002	POLYPROPYLENE FILTER CLOTH 100m²	4	500	1 000	6 000
01-001	STORAGE/MIXING TANKS - VERTICAL	3	1 300	9 700	33 000
	TANK, FLAT BOTTOM. CAPACITY OF 10m³				
	(2.06m ⌀ × 3.00m HEIGHT) C/W				
	MAN HOLE, 3 × 76.2mm FLANGE CONNECTION,				
	LADDER BRACKET AND TOP SUPPORT STRUCTURE				
	FOR TOP MOUNTED AGITATOR; TANK SHOP				
	FABRICATION, FIBERGLASS REINFORCED				
	PLASTIC, [TS-01/03/07]				
TOTAL THIS PAGE					401 000

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER,

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: ZINC PRECIPITATION

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
01-002	STORAGE / MIXING TANKS - VERTICAL				
	TANKS, FLAT BOTTOM, 1 x 50.8mm				
	FLANGE CONNECTION, TOP SUPPORT				
	STRUCTURE FOR TOP MOUNTED AGITATOR;				
	TANK SHOP FABRICATION, FIBERGLASS				
	REINFORCED PLASTIC				
	TANK NO. TS-02 $2m^3$, 1.21m ø x 1.45m	1	1 280	2 800	4 080
	TANK NO TS-04 $1m^3$, 0.96m ø x 1.15m	1	1 280	1 300	2 580
	TANK NO. TS-05 $3m^3$, 1.38m ø x 1.66m	1	1 280	3 900	5 180
	TANK NO. TS-06 $0.02m^3$, 0.26m ø x 0.32m	1	1 280	700	1 980
01-003	TOP CENTER ENTRY AGITATORS, C/W V-BELT				
	DRIVE : GUARD, SHAFT, 55° MAXFLOW				
	1750 RPM MOTOR TEFC 208V FOR 230RPM				
	TURBINE SPEED, ON-TANK RIGID SUPPORT				
	STRUCTURE, WETTED PARTS OF CARBON STEEL				
	AGITATOR NO. MG-02/06, 3 HP MOTOR	2	2 560	5 300	15 720
	AGITATOR NO. MG-01/03/05, 1 HP MOTOR	3	2 560	4 300	20 580
	AGITATOR NO. MG-04, 10Hp MOTOR	1	2 560	4 300	6 860
TOTAL THIS PAGE					56 980

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: ZINC PRECIPITATION

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
01-004	CENTRIFUGAL SINGLE STAGE SOLUTION	4	8100	6100	56800
	PUMP, PUMP HEAD OF 45m AT				
	85m3/hr, PUMP SIZE 4x3x13, C/W				
	DRIVER, 1750 RPM, 25hp MOTOR, AND FRAME				
	[PUMP NO. PB-01/02/03/05]				
01-005	CENTRIFUGAL SINGLE STAGE SOLUTION	4	8100	2740	43360
	PUMP, CS CONSTRUCTION, MECH. SEAL PUMP				
	HEAD OF 25m AT 0.56 m3/hr.				
	PUMP SIZE 1.5x1x6, C/W DRIVER.				
	3600 RPM / 0.25 hp MOTOR AND FRAME				
	[PUMP NO. PB-04/06/07/08]				
01-006	TRANSFER BINS, SIZE TO SUIT UNDER	2	640	1500	4280
	FILTERS APPROXIMATELY 15m³ CAPACITY				
	2m x 3m x 0.25m C/W FORKLIFT				
	LIFT LUGS AND DUMPING CAPABILITY,				
	CARBON STEEL 9.8mm ℄ CONSTRUCTION				
	WITH C4 SUPPORT [TB-01/02]				
TOTAL THIS PAGE					104440

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAPLEACH SOLUTION | PROCESS AREA: ZINC PRECIPITATION

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
07-001	DEAERATION TOWER 0.60 m ø x 6 m,	1	4 320	16 000	20 320
	CARBON STEEL CONSTRUCTION FOR 170 Kpa				
	TO FULL VACUUM OPERATION (-26" Hg),				
	CONSTRUCTED IN TWO FLANGED SECTIONS				
	C/W 3 x 76.2 mm FLANGED CONNECTIONS.				
	DESIGNED FOR VERTICAL OPERATION WITH				
	SUPPORT STRUCTURE. [SQ-01]				
05-002	ZINC ADDITION MIXING CONE 2.3m	1	1 280	3 150	4 430
	DIAMETER C/W 60° SLOPEING SIDES				
	CARBON STEEL CONSTRUCTION. [GS-01]				
02-001	LIQUID RING VACUUM PUMP FOR 28" Hg	1	8 100	4 524	12 624
	VACUUM C/W VACUUM RECEIVER 0.5m³,				
	CARBON STEEL CONSTRUCTION				
01-007	DRY SOLID REAGENT FEEDERS, VOLUMETRIC	3	6 200	13 000	57 600
	SCREW TYPE FEEDER 0.03 TO 50 CUBIC				
	FEET PER HOUR C/W Hopper 1.6 cu ft				
	CAPACITY, AUTO CONTROLS AND ¼ hp MOTOR.				
TOTAL THIS PAGE					94 974

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: ZINC PRECIPITATION

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
15-001	PROCESS BUILDING 10.5m x 16.0m x 6.8m HT.	168m²			136 000
	C/W STRUCTURAL STEEL, FLOOR PAD,				
	INSULATION, CLADDING, FLAT ROOF,				
	PLUMBING, HEATING AND VENTILATION,				
	AND LIGHTING				
15-002	SITE PREPARATION FOR PROCESS BUILDING	1500 m²			15 000
	INCL. CLEARING, GRADING, FOOTING				
	PREPARATION, ROADWAY, AND DRAINAGE				
	SYSTEM.				
15-003	REINFORCED CONCRETE FOUNDATIONS FOR	20.2 m³	20 000	4 000	24 000
	PROCESS EQUIPMENT BASED ON A TOTAL				
	VOLUME OF 20.2 m³ OF FOUNDATIONS.				
	(INCLUDES FRAMEWORK, REBAR AND				
	CONCRETE)				
15-004	PIPING SUPPLY AND INSTALLATION	35m	1 000	1 500	2 500
	OF 25.4 mm (1") PIPE SCH 40 (WELD PIPE) C/W				
	PIPE FITTINGS, HANGERS & SHOP FABRICATION				
TOTAL THIS PAGE					177 500

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: ZINC PRECIPITATION

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
15-005	PIPING SUPPLY AND INSTALLATION	100m	7 500	4 500	12 000
	OF 50.8mm (2") SOLUTION TRANSPORT PIPE				
	SCH 40 C.S. (WELD PIPE) C/W PIPE				
	FITTINGS, HANGERS & SHOP FABRICATION.				
15-005	PIPING SUPPLY AND INSTALLATION OF	25m	4 000	3 000	7 000
	152.4mm (6") VACUUM PIPING SCH 40				
	C.S. (WELD PIPE) C/W PIPE FITTINGS,				
	HANGERS & SHOP FABRICATION				
10-001	INSTRUMENT SUPPLY AND INSTALLATION	6	1 000	6 500	45 000
	C/W CENTRALIZED OPERATOR CONTROL				
	PANEL, WIRING. FOR LEVEL				
	INDICATION AND CONTROL, TURBIDITY				
	INDICATION AND VACUUM INDICATION.				
11-001	ELECTRICAL SUPPLY AND INSTALLATION	100 KVA			45 000
	C/W TRANSFORMER, MOTOR CONTROL				
	SWITCHING, WIRING AND PROTECTION				
	(FOR PROCESS EQUIPMENT).				
TOTAL THIS PAGE					109 000

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: CARBON IN COLUMN

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
07-001	MULTISTAGE CARBON ADSORPTION Column	1	13 200	12 800	26 000
	1.2m Ø x 7.7m, CARBON STEEL				
	CONSTRUCTION FOR ATM. PRESS. OPERATION.				
	CONSTRUCTED IN 3 FLANGED SECTIONS				
	C/W 5 x 76.2mm FLANGED CONNECTIONS				
	AND MAN HOLE. DESIGNED FOR VERTICAL				
	OPERATION WITH SUPPORT STRUCTURE [SQ-01]				
07-002	SINGLE STAGE ELUTION COLUMN, 0.76m Ø x	1	13 200	9 800	23 000
	5.7m, CARBON STEEL CONSTRUCTION				
	FOR 100 Kpa OPERATING PRESSURE,				
	CONSTRUCTED IN 2 FLANGED SECTIONS				
	C/W 5 x 76.2mm FLANGED CONNECTIONS,				
	DESIGNED FOR VERTICAL OPERATION WITH				
	SUPPORT STRUCTURE [SQ-02]				
05-001	TRAP CARBON SCREENS - INCLINED, NON-	2	1 280	2 200	6 960
	VIBRATING, 1m² AREA C/W UNDERFLOW				
	CONE, CARBON STEEL CONSTRUCTION				
TOTAL THIS PAGE					55 960

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCE, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: CARBON IN COLUMN

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
05-002	CARBON SETTLING CONES CAPACITY OF	5	1 260	2 560	19 050
	1.0m³, 2m ø WITH 45° SLOPEING				
	SIDES, CARBON STEEL FABRICATION				
	[GS-01/02/03/04/05]				
01-001	REAGENT TANKS CAPACITY OF 18m³,	2	1 280	13 210	38 980
	2.56m ø x 3.50m C/W 2x 50mm				
	FLANGED CONNECTION, LADDER WELL				
	MANHOLE AND COVER. FLAT TOP				
	AND BOTTOM, CARBON STEEL FABRICATION				
	[TS-01/02]				
01-002	STORAGE/MIXING TANKS - VERTICAL	4	1 280	9 970	45 000
	TANK, FLAT BOTTOM, CAPACITY OF 10m³				
	(2.06m ø x 3.00m) C/W MANHOLE				
	3x 76.2mm FLANGE CONNECTION AND				
	TOP SUPPORT STRUCTURE FOR TOP MOUNTED				
	AGITATOR; TANK SHOP FABRICATION,				
	PONPROPLYENE REINFORCED PLASTIC				
	[TS-03/04/05/06]				
TOTAL THIS PAGE					103 030

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: CARBON IN COLUMN

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
01-003	STORAGE/MIXING TANKS - VERTICAL	2	1 280	3 875	15 550
	TANKS, CAPACITY OF 8m³ (1.38mØ x 1.15m)				
	C/W TOP SUPPORT STRUCTURE FOR				
	AGITATOR MOUNTING, POLYPROPYENE				
	REINFORCED PLASTIC. [TS-07/08]				
01-004	CENTRIFUGAL SINGLE STAGE SOLUTION	3	8 100	4 668	38 304
	PUMP, CARBON STEEL CONSTRUCTION				
	HEAD OF 15m AT A CAPACITY OF				
	65m³/hr, MECH. SEAL PUMP SIZE				
	4x3x8.5 C/W DRIVER, 1750 RPM/7 hp				
	MOTOR AND FRAME [PB-01/02/03]				
01-005	CENTRIFUGAL SINGLE STAGE SOLUTION	3	8 100	2 875	32 925
	PUMP, CARBON STEEL CONSTRUCTION				
	HEAD OF 15m AT A CAPACITY OF				
	3 to 3.5 m³/hr, MECH. SEAL PUMP SIZE				
	1.5x1x8 C/W DRIVER 1750 RPM/				
	0.5 hp MOTOR AND FRAME [PB-04/05/06]				
TOTAL THIS PAGE					86 779

Thibault & Associates Inc. Applied Process Engineering Consultants	CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: CARBON IN Column

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
01-006	CENTRIFUGAL SINGLE STAGE SOLUTION	4	8 100	2 615	42 860
	PUMP, CS CONSTRUCTION, MECH. SEAL				
	PUMP HEAD OF 25m AT 0.51 m³/hr				
	PUMP SIZE 1.5x1x6, C/W DRIVER				
	880RPM / 0.25 HP MOTOR AND FRAME				
	[PUMP NO. PM-01/02/03/04]				
09-001	ROTARY KILN ELECTRICALLY HEATED	1	43 200	86 750	129 950
	SHORT LENGTH (24") / 6" CYLINDER				
	C/W 15.2 KW/hr HEATER (MAX				
	TEMP 2000°F), ½ HP DRIVE AND				
	¼ HP VARIABLE SPEED FEEDER,				
	SUPPORT FRAME, REFRACTORY AND CONTROLS.				
	[EQ-01]				
01-007	ELECTRIC CIRCULATION HEATER	1	720	1 050	1 770
	TYPE NWH, COPPER SHEATH SCH 40				
	CS Body 6KW HEAD LOAD				
	MODEL NWHO-3L-306				
TOTAL THIS PAGE					174 580

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: CARBON IN Column

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
08-001	ELECTROWINNING CELL, 500AMP	2	18 000	16 500	69 000
	8 CATHODE BASKETS PACKED WITH				
	STEEL WOOL, 50mm THICK, 9 ANODES				
	28mm SEPARATION, BASKETS 1.5m²				
	(SOLUTION FLOW PER CELL 1.6 m³/Hr) C/W				
	CONTROLS, TRANSFORMER AND CELL				
	CAPACITY OF 1.6m³ [EE-01/02]				
01-008	TOP CENTER ENTRY AGITATORS, C/W V-BELT				
	DRIVE ⅓ GUARD, SHAFT, TURBINE				
	IMPELLER (SINGLE), C/W 1750RPM				
	MOTOR AND ON TANK SUPPORT				
	STRUCTURE, CS CONSTRUCTION				
	AGITATOR M4-01/02/03 3HP	3	2560	5250	7810
	M4-04 1HP	1	2560	2250	4810
09-002	HEAT EXCHANGER - PLATE TYPE 85m³	1	2560	13 000	15 560
	CARBON STEEL FABRICATION				
TOTAL THIS PAGE					97 180

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: CARBON IN COLUMN

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
01-008	DRY SOLID REAGENT FEEDERS, VOLUMETRIC	2	6 200	13 000	38 400
	SCREW TYPE FEEDER (0.03 TO 50 CUBIC				
	FEET PER HOUR) C/W HOPPER 1.6 CU.M				
	CAPACITY, AUTO CONTROLS AND 1/4 HP MOTOR				
15-001	PROCESS BUILDING 7.3m × 16.0m × 6.8m HT	219m²			195 000
	AND 6.4m × 16.0m × 13.7m HT C/W				
	STRUCTURAL STEEL, FLOOR PAD, INSULATION,				
	CLADDING, FLAT ROOF, PLUMBING, HEATING				
	AND VENTILATION, AND LIGHTING				
15-002	SITE PREPARATION FOR PROCESS BUILDING	2000 m²			20 000
	INCL. CLEARING, GRADING, FOOTING				
	PREPARATION, ROADWAY, AND DRAINAGE				
	SYSTEM.				
TOTAL THIS PAGE					253 400

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CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL | LOCATION: CAPE SPENCER, NEW BRUNSWICK

PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS | PROCESS AREA: CARBON IN Column

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
15-003	REINFORCED CONCRETE FOUNDATIONS FOR PROCESS EQUIPMENT C/W FRAMEWORK, REBAR AND CONCRETE.	24m³	27 000	5 500	32 500
15-004	PIPING, SUPPLY AND INSTALLATION OF 25.4mm (1") PIPE C.S. SCH 40 (WELD PIPE) C/W PIPE FITTINGS, HANGERS & SHOP FABRICATION	150m	5 000	5 500	10 500
15-005	PIPING, SUPPLY AND INSTALLATION OF 50.8mm (2") PIPE C.S. SCH 40 (WELD PIPE) C/W PIPE FITTING, HANGERS & SHOP FABRICATION	95m	7 000	4 000	11 000
15-006	PIPING, SUPPLY AND INSTALLATION OF 50.8 mm (1") PIPE 304 S.S. SCH 40 C/W PIPE FITTINGS, HANGERS & SHOP FABRICATION	35m			5 500
	TOTAL THIS PAGE				59 500

Thibault & Associates Inc.
Applied Process Engineering Consultants

CONCEPTUAL COST ESTIMATION

CLIENT: RESEARCH AND PRODUCTIVITY COUNCIL	LOCATION: CAPE SPENCER, NEW BRUNSWICK
PROJECT: GOLD RECOVERY FROM HEAP LEACH SOLUTIONS	PROCESS AREA: CARBON IN Column

CODE	DESCRIPTION	AMT	UNIT COST LABOR	UNIT COST MAT'L	TOTAL COST
10-001	INSTRUMENTATION SUPPLY AND INSTALLATION	13	1 000	6 500	97 500
	C/W CENTRALIZED OPERATOR CONTROL				
	PANEL, WIRING FOR LEVEL INDICATORS				
	TEMPERATURE CONTROL, FLOW INDICATORS,				
	TEMPERATURE INDICATORS, PH CONTROL				
	AND PRESSURE CONTROL.				
11-001	ELECTRICAL SUPPLY AND INSTALLATION	65 KVA			30 000
	C/W TRANSFORMER, MOTOR CONTROL,				
	SWITCHING, WIRING AND PROTECTION				
TOTAL THIS PAGE					127 500

DATE: MARCH, 1990	BY: D. THIBAULT	PROJECT NO: 6907	REV: 0	PAGE: 8 OF: 8